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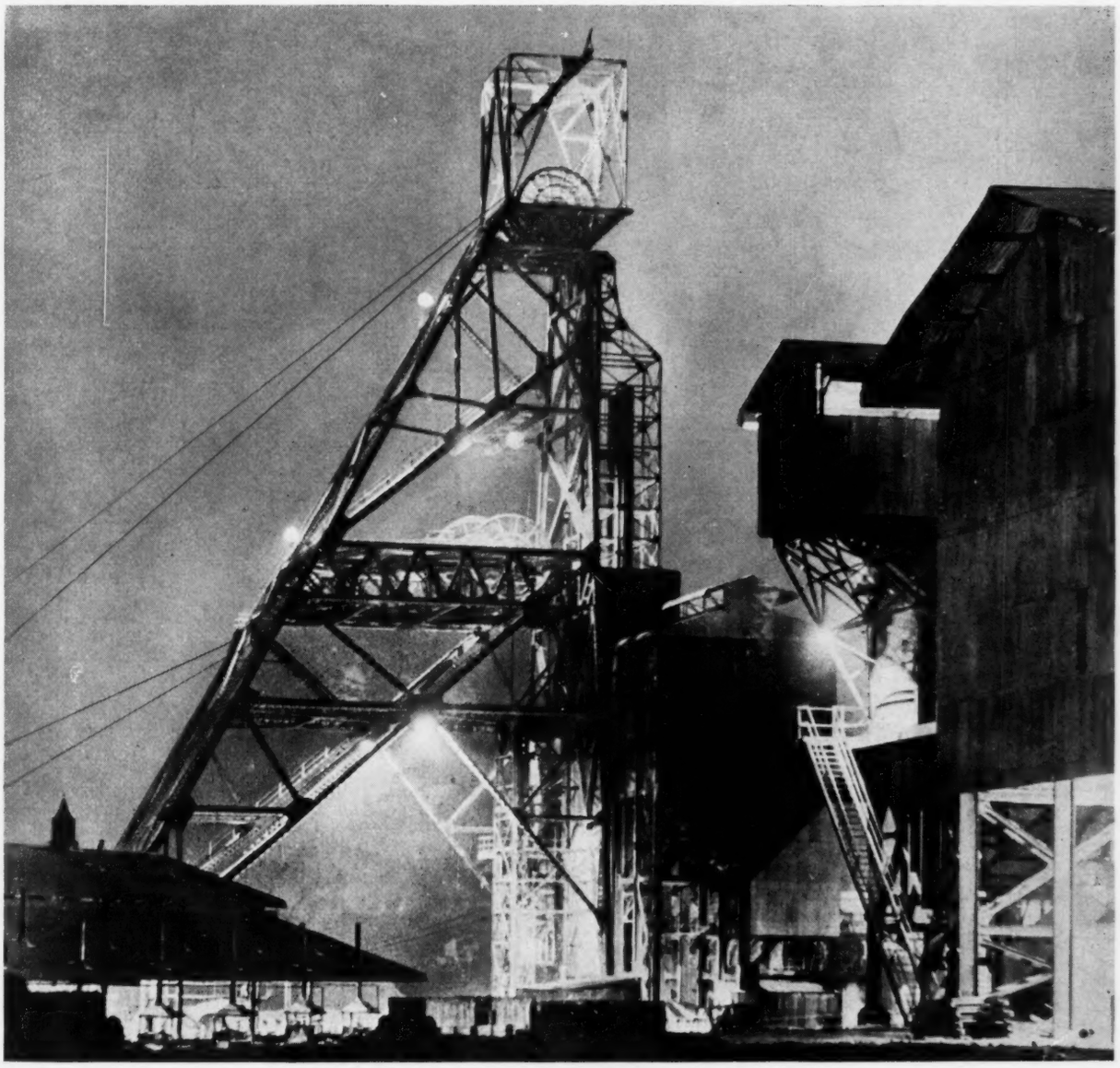

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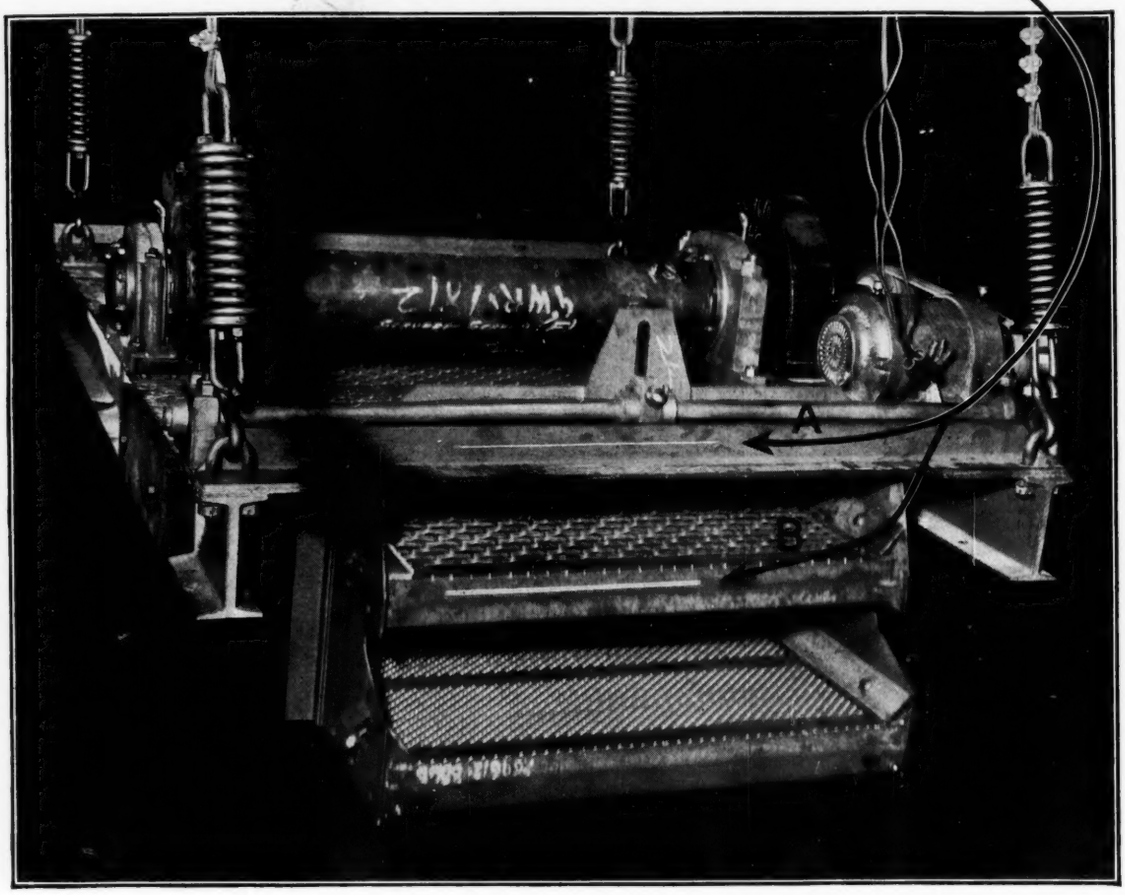

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Engineering and Mining Journal

with which is consolidated
Engineering & Mining World

A. W. ALLEN, *Editor*

Contents for January, 1932

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This copper mine worked from long before the days of Alexander

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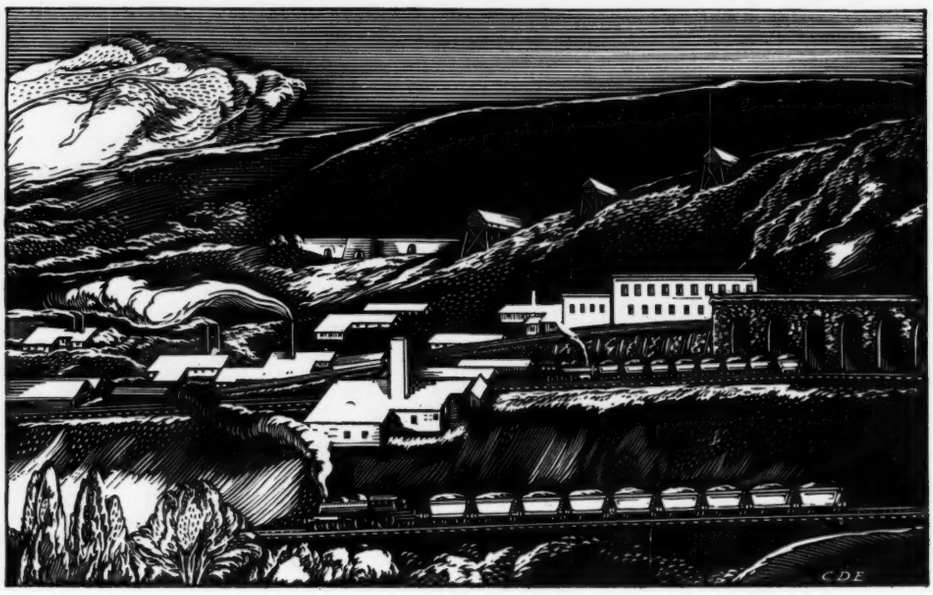

The copper mines of ancient Cyprus were no doubt one of the reasons why this opulent island was fought over, by the armies of Cambyses of Persia, Alexander the Great of Macedon, Harun al Raschid of Arabia. Today this copper mine of Cyprus is using Cyanamid Reagents.

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January, 1932 — Vol. 133—No. 1

A. W. ALLEN, *Editor*

Reducing Costs by Abandoning the Gold Standard

A REPORTED DECISION of the Government of the Union of South Africa to remain on the gold standard, and the proposal of that government to mint a new coin, to be known as the "rand," to replace the pound sterling, is evidence of divergent interests among the widely separated units in the British commonwealth of nations. Should South Africa, as the major world producer of the yellow metal, abandon the gold standard, the belief would be encouraged that adequate precious-metal backing for paper money is not an essential, or at least a bulwark, to financial stability.

The mining industry of the Witwatersrand recently faced this awkward problem. Although desirous of supporting the use of gold as a national reserve, it had to recognize that the countries to which the Union sells products other than gold, including neighboring and distant British colonies, are now operating at a distinct advantage in having thrown in their lot with Great Britain. By subscribing to the policy of inflation that has lowered the gold value of the pound sterling, they have reduced domestic production costs and created what is virtually a tariff wall to protect their own industries. Directors of the Witwatersrand mines, moreover, in considering the future, realize that appreciation in gold output, or even its maintenance at the present level for any considerable time, will depend on the exploitation of the immense reserves of low-grade ore known to exist. And such exploitation is possible only if production costs can be lowered. The industry sees that such reduction can be easily and speedily effected by following the example of Great Britain and abandoning the gold standard. The Transvaal Chamber of Mines, through its Gold Producers' Committee, estimates that a ten per cent devaluation of Union currency would automatically effect a reduction of over two shillings per ton in working costs, by reason of the higher sale price of gold in terms of the new currency. This assumes, of course, that wages and commodity prices are not raised—a tendency that is already noticeable in regard to commodity prices in England.

Opportunity to make so marked a reduction in working costs is of wide significance to the gold industry of the Transvaal. A recent investigation, instigated by the Low-Grade Ore Commission, has indicated that an economy of two shillings per ton milled would augment the existing ore reserves of the Witwatersrand to an extent that would prolong the life of the mines by fifty per cent. Additional expenditures made by the mines under such conditions may be estimated at £150,000,000, presumably in terms of the devaluated currency. So significant and tempting an opportunity rarely comes to the directors of any great group enterprise. The Transvaal Chamber of Mines apparently made every attempt to justify recommendation in favor of a continued adherence to the gold standard in the Union of South Africa, but its own interests and the interests of South African producers of agricultural and other commodities dictated a recommendation in favor of a devaluated

Historical Summary

Engineering and Mining Journal was founded in 1866 as the *American Journal of Mining*, the title being changed in 1869. Publications absorbed comprise: *Coal and Iron Record* (1875); *The Mining Review* (1877); *The Polytechnic Review* (1879); *Mining and Metallurgy* (1902); *The Mining Magazine* (1906); *Mining and Engineering World* (1917); *Engineering and Mining World* (1932).

The following have served as editors in chief of *Engineering and Mining Journal* and its predecessor publication: G. F. Dawson (1866-67); R. W. Raymond (1867-74); R. W. Raymond and R. P. Rothwell (1874-89); R. P. Rothwell (1890-1901); D. T. Day (1901-02); T. A. Rickard (1903-05); W. R. Ingalls (1905-19); J. E. Spurr (1919-27); A. W. Allen (1927-).

Engineering and Mining Journal was consolidated with *Mining and Scientific Press* in 1922, as *Engineering and Mining Journal-Press.* The title was changed to *Engineering and Mining Journal* in 1926.

The editorial staff of *Engineering and Mining Journal* comprises: A. W. Allen, editor; George J. Young, A. H. Hubbell, H. D. Keiser, associate editors; W. N. P. Reed, S. D. Strauss, J. B. Huttl, B. H. Strom, assistant editors; H. H. Wanders, market editor.

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sterling. The government, however, decided otherwise.

The history of mining is replete with examples of cost reductions that have enabled ore of extremely low grade to be treated at a profit. Copper metallurgists and mining engineers have performed miracles of achievement in this connection. The problem of treating the lower-grade Witwatersrand ores at a profit, assuming continuance of the present gold standard currency, can and will be solved. Better to attack the problem from the technical and engineering standpoint than to rely on a currency debasement the effect of which cannot be permanent, and which, if carried to its logical conclusion, might deprive the Witwatersrand industry of a market for the product of its mines.

Copper Industry Attacks Its Problems

ANOTHER important curtailment of world copper production is to be instituted this month as the result of the recent conferences of representatives of all but about ten per cent of the total productive capacity. Just a little over a year ago, when copper was selling at nine and a half cents per pound, a similar conference resulted in agreement on a fifteen per cent cut, amounting to about 20,000 tons per annum. Late last June, after copper had dropped to about eight cents, and too much was still being produced, another effort at general curtailment was made, but without success. However, with copper at the all-time low of six and a half cents or less, as it was in December, and with stocks increasing faster than ever, necessity for drastic action was such that producers finally agreed again on a general reduction of output of about 35,000 tons per annum.

Inspection of the official announcement of the producers reveals that extreme care was taken to justify their action on legal grounds, even though restriction of production leads to an advance in price, which, in fact, is implicitly admitted as one of the aims. No formal agreement was made and "there is no authority to enforce the reduced rate of production." The agreement automatically ends when stocks fall below four months' requirements, when the price exceeds twelve cents, or in any case at the end of a year. These limitations should effectively disarm the criticism of even the most ardent believer in the anti-trust laws of the United States. No action is likely to be taken against any participant in the agreement, or investigation made. If an industry cannot even go thus far in the effort to conduct its business without loss, no sensible person can gainsay the contention that the Sherman Act should be amended.

One phase of the agreement, given attention by the *American Metal Market,* is the refusal by one, or perhaps more, of the important producers to release data on copper production, shipments, and stocks. The life of the agreement may depend on the current position of shipments and stocks, so one may assume that these will become known in due time. No announcement of the reason for withholding data for the last two months has been given. If it be in an effort to insure more complete statistics, including data on the position of consumers as well as producers, the effort is deserving of success. If, on the other hand, the motivation arises from a desire to suppress bad news, or because of a belief that data on the copper industry are not the public's business, the action taken is indefensible. Bad news is seldom so bad in fact as in imagination; and the investing public that helped to finance the copper industry, that now finds—comparing 1929 highs with recent lows—its investments worth as little as five per cent of former value, has a right to as full information on production and stocks now as it had in the past.

Looking to the future, one must not assume that the present low condition of the copper industry is peculiar to itself, among the other metal industries, or that African production will prevent the present generation of producers in other parts of the world from getting back on their feet. Copper, at its recent low, was selling at about 43 per cent of the average of the past ten years. The same could be said of tin and silver. Zinc was not much better off, at 51 per cent; and lead, even in the tariff-protected market of the United States, at 55 per cent. Production of African copper at any price below ten cents will not be rapid, nor is planned output there sufficient to warrant the belief that copper from other established mines will not be required in the next few years to meet world demand. The peak of copper consumption—2,091,000 tons in 1929—is likely to be exceeded within the next few years; and production from African mines, even though of comparatively low cost, may well prove to be no more of a thorn in the sides of other producers than has the Chilean production in the last fifteen years. That, too, has been low-cost copper from enormous ore reserves.

New Roads to Recovery By Increasing Opportunity

The extent of Alaska's minerals and other natural resources would, in normal times, justify rapid development of the Territory, although to stimulate such development now, when an oversupply of most commodities exists, disregarding gold, might be considered unjustifiable. This view is a short-sighted one, however, because time would be required to effect a marked increase in output in any line, save at the fish canneries —already well established.

Alaska, with its 600,000 square miles of land, could support many times its present population. Much of its vast area is mountainous and forbidding, harsh in climate; but this is also true of Scandinavia, which, about one-half the size of Alaska, supports nearly ten million people. The government at Washington could well encourage the settling of Alaska on a more extensive scale. Chance to gain a mere living, under circumstances where one's efforts would insure this as a reward, would attract many of the sturdier sort who have become discouraged at the imperfect distribution of the fruits of an industrial civilization.

Alaska apparently is being conserved for the needs of a future generation. At least, this was the idea when conservation measures were adopted a decade ago. That generation is now here. An influx of population to the Territory from the United States would stimulate business on the Pacific Coast; it might prove the turning of the tide. Other changes inland would be induced, leading to a speedier recovery from the depression, which by many is held to be largely psychological and rooted

in fear. As the mind of man fell into fear, so will it emerge; and the cause of the emergence may be as intangible and indefinable as that which precipitated the plunge.

Prerequisite to the development of Alaska is the provision of roads, without which the Territory is uninviting to a would-be settler. Road building there is a slow and expensive process. The Territory's Road Commission, for the year ended June 30, 1930, was charged with the maintenance of 1,342 miles of road, 85 miles of tramway, 905 miles of sled road, 4,277 miles of permanent trail and 314 miles of temporarily "flagged" trail, as well as 736 miles of telephone lines, 29 air landing fields, and 40 shelter cabins. For its road work, including maintenance of existing roads as well as new construction, it is allowed about $1,080,000 annually. Of this, 75 per cent is required for maintenance and improvement. Thus only $250,000 annually remains for building new roads in a country almost three times the size of France. Improved and extended means of transportation would stimulate gold mining in Alaska, providing livelihood for many who are now unemployed.

Germany Provides a Lesson in Regulation

THE POTASH INDUSTRY in Germany affords an interesting chapter in the mining history of the world. From its inception in 1857, when the full value of potash as a fertilizer was recognized, the industry expanded rapidly, until, in 1910, productive capacity so exceeded consumption that the Imperial German Government was compelled to apply regulatory measures. Monopolistic control of the world's potash markets ceased with the War, at the close of which the Alsatian mines passed under the control of France. Impending disaster to the German industry then led to further participation in it by the government. This governmental control continued and is now particularly active.

In his article "Potash Mining in Germany," which appears in this issue, Prof. Ernest R. Lilley, of the department of geology at New York University, recounts the historical facts in detail and presents an informative description of the geology of the area, as well as technical data concerning the mining of the deposits. He shows that the government of Germany is primarily concerned in preventing the exploitation of new mines and in reducing output from those already developed. Estimates, he says, indicate that Germany, possessing high-grade potash ore in excess of that of all other countries combined, could supply the world's needs for the next thousand years, at the present rate of consumption, without seriously depleting its reserves.

In these days of intensive nationalism and towering tariffs, however, the question is not so much what a nation can do to supply the world as what the other nations will allow it to do. Despite the vast stores of potash minerals in Germany, the search continues elsewhere for reserves that will insure independence when a national emergency arises, and employment for workers at all times in meeting normal domestic needs. Remarkable progress, for instance, has been made by the United States Potash Company in developing the carnallite, polyhalite, and sylvite deposits at Carlsbad, New Mexico, in connection with which advantage can be taken of the most economical, most efficient, and most recent developments in mining methods and equipment design, and the best available technique for the recovery of a soluble salt from an insoluble gangue, and its purification on a large scale.

A study of the recent history of the German potash industry provides pertinent and useful information that should aid in solving the problems of overproduction and excess capacity that face most of the other branches of mining throughout the world. In this connection, Professor Lilley's introductory article is particularly timely and suggestive.

The Gospel of "It Can Be Done"

NECESSITY has ever been a spur to the exercise of ingenuity. When hard times threaten earnings, salaries, and positions, the able and efficient among a company's employees feel the urge to demonstrate ability and efficiency; and those in whom these qualities have been dormant are influenced to display them. This response, natural to men of the better sort, is not necessarily the result merely of loyalty or fear. Only in the mentally sluggish and dull-witted do world conditions like the present fail to arouse a desire to overcome obstacles, to rise to new achievement. Only among dullards is the reaction limited to fear.

Plunged in the deep trough of an unusual depression, the mining industry has afforded several splendid examples of dogged perseverance. The technical staffs of companies have responded well to the new conditions, proving that the seemingly impossible could be accomplished if only the goal were visualized, effort put forth, and money spent where such spending was imperative. But the need for financial disbursement has occasionally proved a stumbling block. "What!" say some, "Spend money while stock prices are where they are!" The thought appears to dismay a type of executive or director who, far removed from the locale of mine or plant, is usually more familiar with financial affairs than with men and machines. Such a man needs more faith in the gospel of "It can be done."

To understand this gospel one must first take note of past and contemporary technical achievement. Every director should know of the company that, in the depression of 1921, although at first convinced that underground costs could not be reduced, succeeded in cutting them in two. He should consider the copper company in Utah, which, since the depression began, has lowered production cost materially by electrifying its haulage, at great expense—but at one that gives a handsome return. With such knowledge he may be more receptive to ideas advanced by his subordinates. Thus will he begin to see a way out of the slough of despond. If he be one of the many who lost in the stock-market decline—and, in consequence, has become hesitant about investing—he should remember that an expenditure to modernize plant or process is no gamble if it yields a return as big as or bigger than the money could earn if placed at interest. Of one thing he may be sure: a favorable attitude toward the recommendations of the operating staff will be repaid many times in added zeal, enthusiasm, and loyalty.

Free to go ahead in putting new ideas and plans to the test, the men "on the ground" are almost sure to make a significant contribution to lower costs and improved technical efficiency. The news should then be broadcast, as a means of stimulating others to similar achievement. Mr. A. H. Hubbell's article in this issue will serve as the opening gun in such a campaign. Each company and each responsible official can contribute to hasten the day when the mining industry emerges from the doldrums.

Asbestos Industry Faces Intensive Competition

THE world asbestos industry is entering a new era of competition. Producers are in two groups—the Rhodesian-Canadian and the Russian. Rapid strides are being made in the U.S.S.R. in the expectation of an output in 1933 of a quarter of a million tons, equal to the Canadian output in 1928 in quantity; and, because of the length of the Russian fiber, equal to the total world output of that year in intrinsic value. Russia plans to export about one-third of its production. Sharp competition for world markets is therefore a certainty, the trade eventually going to the low-cost producers. Cheap asbestos will find a ready market, and the buyer will not be unduly influenced by comparison of labor and living conditions in Canada, Africa, and Russia. Even Soviet gasoline—petrol—eventually displaced other brands in England because of its low price.

The two major asbestos producing regions of the world are Canada and Russia; and *Engineering and Mining Journal* is fortunate in being able to present to its readers a series of articles, the first of which appears in this issue, by Mr. Walter RuKeyser, who has had a wide experience in the mining and processing of asbestos in both countries. Opportunity is thus provided to outline the inevitable and necessary trends toward centralized operations, the mechanization of plant, and a pooling of technical knowledge. The first article describes fundamental changes in mining practice that characterize the recent history of the Canadian industry. Details of mill practices in Quebec, and flowsheet modifications, will be discussed in a second article, to be followed by complete technical data in regard to the exploitation of the asbestos mines of Russia.

The release of so much engineering information should encourage the emulation of sound technical practice; provide information helpful to the promoters of asbestos enterprises in other parts of the world, and hasten those economies in mine and plant that must be put into practice by major producers if they are to survive the intensive competition for world markets that is impending.

Australian Gold Mining on the Upgrade

AUSTRALIA, far from the centers of Occidental civilization, was first developed largely because of the white man's quest for gold, the metal that lured him to settle in Victoria in the middle of the nineteenth century, and spurred him to open up New South Wales, Queensland, and, finally, Western Australia. The greater part of Australia is still thinly peopled. Huge tracts of land in the central and northern areas are unsettled wastes. If these regions, some day, are to attract and support a considerable population, that prospect is based largely on the mineral resources that may be developed there. And, as is usually the case in areas far removed from cheap transportation, gold is the mineral resource that will receive first attention.

Australia, in the eighty years since the first important gold finds were made, has grown beyond the stature of a nation dependent on one industry. Its industrial and political leaders have been ambitious. Desiring to make Australia an industrial nation, they embarked on a policy of fostering domestic industries by tariffs and other aids. Costs have risen, and, with them, the standard of living and the demands of the workers.

Under such conditions, the industries that depended on a world market in which to sell their produce were severely handicapped. Among these, of course, was gold. Australia produced only one-seventh of the gold in the five years from 1926 to 1930 that it did in the five years from 1901 to 1905. The most important reason for that decline was the handicap that tariffs and high-wage schedules imposed on the gold mines.

The depression brought the Australian governments, Federal and State, to dire straits. Basic commodity-producing industries have been hard hit. Domestic manufacturing industries declined also. So, for the time being at least, Australia must turn from its thoughts of industrial ascendancy to the metal, gold, which alone commands a stable price and a ready market. And Australia still has much gold in the ground. Though the rich surface deposits may be exhausted, immense reserves of low-grade material exist, which have not been touched because of high production costs.

In this issue a correspondent describes the status of gold mining in Western Australia, the premier gold-producing State. His prediction of an output of 600,000 ounces in 1932 appears to be justified, for 1931 will probably record a production of 500,000 ounces, or 80,000 ounces more than 1930. The increase comes as a result of several stimuli. New deposits have been developed. Improved practice is resulting from the installation of modern, large-scale equipment. A Federal bounty on excess production over the average of 1927-28-29 will be paid in 1932. The depreciation in Australian currency has added about 35 per cent to the price the mines obtain (in that currency) for their output. Both bounty and premium, however, are artificial aids. Because they are likely, at any time, to disappear, the growth they stimulate is not a healthy one. Australian gold-mining companies, by taking advantage of the present low cost of supplies and of development work, should be able to succeed without subsidies, on which too much dependence in the future would be unwise.

Economies and Efficiencies In Mine and Plant

A. H. Hubbell
Associate Editor

MOST COMPANIES today find earnings greatly reduced or even a negative quantity, owing to inability to market their products at a price that permits a profit. To remedy this situation they can seek the following objectives: (1) Curtailment of production, thereby to cut loss and improve price; (2) merger with other producers, thereby to reduce overhead expenses and better enable themselves to control the situation; (3) reduction of operating costs by increased efficiency; (4) treatment of low-cost tailing residues and utilization of otherwise wasted products, either current or accumulated.

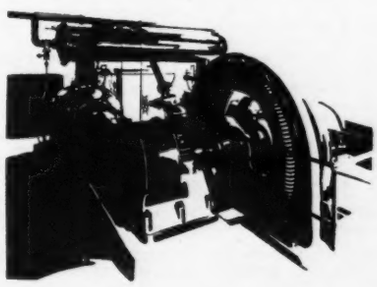
Whatever program of merger or curtailment be adopted by the directors of a company, the operating man at quarry, mine, or treatment plant, under pressure to make a better showing, is concerned chiefly with matters of cost reduction, avoidance of waste, utilization of byproducts, and possible savings of every kind. This is well illustrated by a comment of Dr. Oliver Bowles in reviewing the non-metallic mineral industries in these pages last February. He said: "An important effect of the depression is the interest aroused in cost reduction. As a rule, this has not taken form in wage reduction but rather in waste elimination, in better utilization of materials, and in technologic improvements in concentration and treatment." For the benefit of the operating man this article brings together a wide variety of cases, in each of which has been effected some economy or improvement in efficiency of operation, many of which are susceptible of being expressed in terms of dollars and cents. Such specific illustrations of practice outweigh any abstract preachment.

Success breeds competition and emulation. Practices resulting in greater efficiency of operation are usually quickly adopted elsewhere to the extent permitted by conditions, as they become known, and are often improved upon. An example is flotation. Another is the use of anti-friction bearings. And at any time a survey of the various divisions of the mining industry will show certain practices in widespread if not universal use, others that have gained only partial acceptance, perhaps because they are insufficiently known, and still others that up to the moment have secured but a limited foothold, doubtless for a similar reason. Indirectly, this is significant of the desirability of exchanging technical information.

Reduction of unit costs by increasing the scale of operations is denied to many companies because of the need for curtailing output. Exceptions are those companies producing gold, this for obvious reasons; and the producers of other metals and mineral products who operate several mines or properties, some of which can be shut down, with an accompanying increase in tonnage and efficiency at the others. The necessity of keeping men employed has prevented as widespread an adoption of this policy as facts might occasionally seem to have required. A median course is found where there are two mines of the same capacity, both running half the time; here a higher efficiency and lower costs can probably be obtained with little or no lessening of employment, by shutting one mine down and running the other at capacity.

No humanitarian injunction stands in the way of cutting costs through improving process, method or equipment. Even where this results in labor saving, it is likely to be tolerated because no extensive layoff of employees is involved, so largely mechanized already are the mining and treatment of ore and rock.

Akin to cost reduction, and directly contributing to it, is waste elimination. Waste may be of varied kinds. Anything of value is wasted to some extent if used without giving the full service of which it is capable. Among wastes may be power, heat, steam, smelter gases, mill dust containing values, and metal in equipment. Losses also ensue when the life of equipment is shortened by abrasion, corrosion, and lack of proper lubrication; when lubricant itself is wasted by careless use or wrong selection; when supplies in general lose in value by negligent warehousing, distributing, and handling; and when overhead charges are incurred and increased on equipment standing idle because of breakdown or other reasons.



Likewise, carelessness in employee relationships may result in waste, as when money invested in training men is lost because of labor turnover, from which few companies suffer at the moment. Failure to provide safe working conditions means loss of working time from accidents, and the possibility of damage suits. "Safety pays," according to the slogan. However, the purpose of this article is not to dwell on such relatively intangible sources of waste.

Waste also arises from failure to utilize byproducts. So commonly have these been wasted by force of circumstances that many an operator has been slow to see in them a possible source of profit. Indeed, they have not been dignified usually by the term "byproduct." But among operators are shining examples of the opposite sort. One of them, situated over a hundred miles from any large community, is proud of the fact that everything in his ore is, or will be, turned to profit—copper, zinc, sulphur, iron sinter, smelter slag, and flotation tailing.

Tailings or residues, accumulated in past years and containing appreciable quantities of the constituent for which the ore was originally treated, but which was imperfectly removed because of relatively crude or faulty treatment processes, afford the promise of a profit today in numerous localities. Examples are the huge dumps of the gold-bearing sands in the Transvaal, and the accumulations of sandy clays and ochers that are to be found in many a china clay and ocher region, in the United States and abroad, because the crude method used for settling the sand out of the clay or ocher could not remove all of the clay or ocher from the sand.

Facilitating the spread of new ideas that tend to lead to greater efficiency in operation is the fact that of the functions involved in mining and in treating ores and mineral products in the mining industry (aside from fuels, here arbitrarily excluded from consideration), over 80 per cent are common to

both the metal and non-metallic divisions of the industry. To demonstrate this graphically the accompanying chart has been prepared. The functional operations, all of them requiring some kind of equipment for their performance, are shown at the top of the chart, as headings of the respective columns. Right and left are shown the products of the mining industry (except fuels), in ten groups for convenience, five of the groups metallic and five non-metallic. In each column, under the respective functional heading, is indicated by means of a dot opposite each mineral group whether or not that function is involved in the production of that particular group. With this explanation, the chart will speak for itself. Not only is it at once apparent that most of the functions shown as column headings are involved in producing at least some of the mineral groups in each of the two major divisions, metal and non-metallic; but, in addition, one notes that fully half of the functions are involved in the production of all ten groups. A striking demonstration this of the truth that the various groups of the mining industries have much in common and that each may find it worth while to study the practice of the others. For this reason, the instances of cost reduction, waste elimination, and plant economies to be cited have a significance for all operators, irrespective of the minerals mined.

Many factors influence mining costs. Sometimes the costs can be reduced by modifying a single factor. In other operations, an appreciable saving can be made only by pursuing a definite campaign having many angles. An example of the first sort is afforded at the Mount Hope iron mine, in New Jersey, where, in shrinkage stoping, the tonnage broken per man-shift was raised from 15.5 to over 60, with corresponding reduction in cost, simply by substituting for the ordinary stoper a hammer drill equipped with an air-feed stoper leg, and mounting it on a board so as to drill 16-ft. rounds at angles of 30 deg. or less to the horizontal, instead of putting in 5-ft. holes at steep angles in the back in the usual fashion. In contrast with this is the cost-cutting campaign on the Central Rand, South Africa, whereby mining costs were cut from 25s. 8d. in 1921 to 19s. 7d. in 1929. The increase in output per man that yielded this result was achieved by increasing the total hours worked 20 per cent without altering the hours per man, this through better discipline; by substituting a piece-work system for the old contract system, with a bonus for efficiency; by standardizing on a certain type of jackhammer and drilling only one hole per bench, which was pointed by a device; by reducing the size of steel; and, capping all, by conducting extensive research. Here are two cases, one involving white labor and the other native blacks supervised by whites, both cases having a significance for operators in other metallic operations as well as those producing non-metallics. Adequate supervision has an important effect on costs where native labor is employed. Conscientious white supervision on a generous scale is necessary to secure efficiency, according to the manager, Russell B. Woakes, of the Indian Copper Corporation, in India. An increased supervisory staff has immediately resulted in reduced working costs at this company's mine, and in improving the grade of ore extracted.

Drill-steel sharpening presents an opportunity for reducing costs at many mines, in some by the centralizing of the work for several adjacent mines, but more frequently by improving the efficiency and quality of the furnace work. At the Homestake mine, in South Dakota, the adoption of natural gas in place of distillate in the sharpening furnaces reduced the fuel cost per bit shaped and hardened from $0.00844 to $0.00271 in 1930, or a total saving of $5,832 in that year. In this instance the excellence of the apparatus used also played a part. Sharpening practice was likewise improved at the central shop of the Copper Queen, in Arizona, by installing new oil-fired furnaces in which the use of 12-lb. steam instead of air for combustion gives a moist reducing flame that keeps

Technical Interrelation of the Metal and Non-Metal

| | Exploration | Mining and quarrying | | |

the steel from carbonizing and scaling while heating. Economies in oil and air were thus effected.

After repeated sharpening, normalizing of drill steel, by bringing it to its decalescence point to restore its normal texture, has been found profitable—by one mine at least. This was on the Rand, in South Africa, the company adopting the practice being reported to have thereby reduced its purchases of new drill steel by about 90 per cent and the breakage to approximately 15 per cent of its former figure. The first cost of the normalizing plant was heavy, it is said; but after allowing for working costs in labor and current, the capital outlay was redeemed in three months by the reduction in the cost of steel purchased.

Where auger steel is used, improved results in sharpening may likewise be obtained, as at the Montreal iron mine, in Wisconsin, where, by using a special shaping die to get the proper auger twist, the output per sharpener was raised from 25 to 35 per hour and higher.

In core drilling and so-called deep-hole drilling, costs have been lowered interestingly. In two separate instances, light, compact, and exceedingly mobile diamond drills have been developed, with which, by reason of the greater convenience and economy attendant upon their use, drilling costs ranging from 53c. to 82c. per foot have been attained for holes up to 500 ft. deep but mostly less than 200 ft. Surpassing this, however, has been the lessening of the expense for diamonds by the new practice of setting bits with a large number of small, uniformly sized South African bort instead of with a relatively few large carbons. Even allowing for the greater number of the small stones, their use has materially lowered the cost of stones per foot of hole, in addition to decreasing the capital outlay required. Further conducive to lowered costs is the mine operator's ability today to purchase bits completely set with small bort, ready for service.

In test drilling, as in other lines, the hard-facing of the edge of the cutting tool, or bit, has given greatly increased life. In such manner stelliting has been used recently in deep-hole drilling at Gilman, Colo., the bit cutting six to eight times as much rock as would an ordinary bit of tempered steel, and in some instances drilling up to 125 ft. of hole before losing gage. Many holes are bottomed with the original bit. In five months in 1931, deep-hole drilling costs averaging $0.958 per foot were obtained, and even lower than this in the six years that preceded.

Much research has been done with the purpose of devising cheaper means of effective ground support. Schaefer lining, consisting of an arch formed of concrete blocks with creosoted yellow-pine blocks at intervals to take up the movement of the roof, have been tried in a coal mine in Pennsylvania—with reported success. Comparisons are difficult, owing to varying conditions. In one section of the mine the cost of replacing a yard of timbering with untreated timber was found to be $10.50; with Schaefer lining $15.50; and with treated timber, $22.50. This indicated a decided advantage for the lining as compared with treated timber. On the Rand the relative costs of reef packing, waste packing, concrete pillars, sand filling, and other means have been carefully investigated.

Sand filling—that is, classified mill tailing, pumped underground into stopes through rubber-lined pipes—has proved to be an economical method of ground support at the Matahambre mines, in Cuba, since February, 1929. Actual cost per ton placed in stopes is reported not to exceed 20c. Mining cost per ton has been reduced by more than $1, owing to the convenience and economy of this method and the greater production made possible. Opportunity for cheaper mining elsewhere is thought to lie in the modification of mining method permitting use of some such effective and economical means of stope filling, now that the problem of pipe abrasion has been overcome.

Scraping has been responsible for marked reductions in underground mining costs. During the past year the Copper Range Company, Painesdale,

Industries Through Common Production and Extraction Processes

Preliminary, intermediate and final treatment of raw materials — General functions at treatment plants — Auxiliary engineering activities

42	43	44	45	46	47	48	49	50	51	52	53	54	55	56	57	58	59	60	61	62	63	64	65	66	67	68	69	70	71	72	73	74	75	76	77	78	79	80	81	82	83	
Settling	Flotation	Magnetic separation, also 35	Electrostatic separation	Jigging	Tabling	Drying	Packing	Amalgamation	Leaching	Agitation	Washing	Filtering	Precipitation	Roasting, calcining	Sintering	Smelting	Volatilization, distilling	Refining, also 61	Refining, electrolytic, also 60	Fuel preparation	Flux preparation	Tailing and slag disposal	Fume and dust recovery, also 34	Air compressing, also 7	Hearting	Lighting, also 46	Sampling	Laboratory	Power generation	Power transmission, also 16	Power application (motors)	Process steam	Surveying	Plant design and construction	Shop work	Water supply	Fire prevention	Safety work	Housing	Railway, highway, tramway building, also 12, 29	Office work	
•	•	•		•	•	•	•		•	•	•	•	•	•	•	•	•	•	•	•	•	•	•	•	•	•	•	•	•	•		•	•	•	•	•	•	•	•	•	Copper
•	•			•	•	•	•	•	•	•	•	•	•		•		•	•	•	•	•	•	•	•	•	•	•	•	•		•	•	•	•	•	•	•	•	•	Gold and Silver
•	•	•	•	•	•	•	•		•	•	•	•	•	•	•	•	•	•	•	•	•	•	•	•	•	•	•	•	•		•	•	•	•	•	•	•	•	•	Lead and Zinc
•	•	•		•	•						•				•	•			•	•	•	•		•	•	•	•	•	•		•	•	•	•	•	•	•	•	•	Iron Ore
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(2) Among the more important non-metallic industries in this group are included: Asbestos, barite, diamonds, feldspar, kaolin, fluorspar, fullers' earth, graphite, magnesite, mica, nitrates, phosphate rock, potash, pyrites, salt.

Mich., applied it to mucking in shaft sinking for the first time on record. In contrast with its former standard of 9 to 10 tons per man shift for hand mucking in a shaft, the company is now getting 19 to 21 tons with the bucket dump 330 ft. above the bottom. With a hoisting distance not exceeding 200 ft., it hopes to get as high as 40 tons. Another interesting instance is found at the Montreal mine, already mentioned, where, in connection with top slicing, chutes and cars have been abolished in the transfer drifts and scraping substituted, with a consequent trebling of the tonnage handled per man engaged in transport and other marked advantages. Incident, moreover, to the increased mechanization of diatomite mining in California, where open-pit extraction is giving way to underground work, the diatomite is being handled in stopes by air-operated scraper hoists.

Underground conveying under certain conditions has advantages long recognized abroad. In 1927, the Boleo company in Lower California, Mexico, replaced its cars with conveyors, thereby increasing the output per man-shift from 1,352 kg. in that year to 1,916 kg. in 1930, and to 2,126 kg. in March, 1931. The cost of stoping per square meter was thus reduced to 3.59 Mexican pesos from 6.81 pesos.

Where mine cars are used for transport, costs can often be lowered appreciably by decreasing the cost of maintenance, either through improvement in design, as, for instance, the adoption of anti-friction bearings, or by substituting a type of car better suited to the work. At the Mount Hope mine, previously mentioned, both of these changes were adopted to advantage: Granby-type cars, equipped with anti-friction bearings, were substituted for gable-bottom cars using plain bearings, the repair cost thereby being reduced from $0.044 per ton handled in 1925 to $0.011 in 1928-29—a saving of 3c.

Electrification of ore-transport systems, whether on surface or underground, has usually resulted in important cost reductions. Utah Copper, by adopting electric locomotives on its 65-mile system, increased the number of cars per train from eight to twelve, and the speed 25 per cent; simplified its traffic problem; and increased the per cent of capacity at which the shovels worked. The first installation required two 75-ton electric locomotives with 7 miles of catenary overhead, and paid for itself out of savings in eighteen months. Construction work involved in changing over from steam to electric haulage began in 1928 and was completed by the end of 1929. The cost of the electrical energy carried in the overhead system at the mines is said to be less today than the cost of the water used by the steam locomotives.

The savings accruing from this improvement are indicated in Table I by the comparison (furnished by the company) of electric haulage costs for 1930 and 1931 with the best steam haulage costs as represented by the average costs for 1927. Adjusted to equal stripping ratio of waste to ore, the actual saving per ton of ore mined is 2.80c., which represents a decrease of 34.3 per cent compared with steam haulage.

In erecting this 65-mile system, economies were effected by applying heavy electric traction principles to ore haulage. Catenary overhead was used in the classification and crusher yards, direct suspension on mine benches and switching tracks, and special offset construction on portable sections and new waste dumps. Last year a long section of the initial installation was taken down, tracks were changed and realigned into the crushers, and the same materials put back into service, without expenditure for new materials.

Similarly, economies from lower maintenance costs have been made in improving trolley suspension underground. At its mines at Gypsum, Ohio, the United States Gypsum Company electrified its main haulageway so as to do away with slack sections in the overhead system. After six years of operation, tension has been maintained, the 4/0 grooved hard-drawn copper wire used shows slight wear, and no perceptible pounding of wire at the clamps has been detected.

Many opportunities still exist for improving ore haulage, especially intermediate haulage, underground. By modernizing obsolete haulage practice, a Birmingham iron-ore producer, whose haulage costs constituted 25 per cent of production costs, reduced this excessive ratio to 16 per cent. Likewise, at Butte, large savings have been effected by electrifying haulage underground. Three small storage-battery locomotives, installed by United Verde Extension, Jerome, Ariz., paid for themselves in two years, saving, as they did, the labor cost of six to eight men per locomotive. They are also reported to have cut down the clean-up time required for development and drift headings, facilitated train movement, and increased production by improving the efficiency on the main haulage lines. At Lyons, Kan., the Western Rock Salt Company found that each storage-battery tramming locomotive paid for itself in 1½ years, replacing four men and improving morale throughout the mine. In Cuba, the Matahambre company reports that four such locomotives underground and one on surface had saved six men each.

Open-pit transport system can often be remodeled to advantage, so as to eliminate the defects arising from haphazard growth. In recent years, a limestone company in Missouri has remodeled its plant and pit transportation layout so as to make it more efficient, incidentally standardizing all track supplies. Average cost of transportation, including delivery and dumping of rock into the top of the kilns, was 14c. per ton for the last eleven months of 1931, compared with 22.5c. for the nine months preceding the major change in the transportation system in 1927. The 14c. figure was obtained on a tonnage 11 per cent lower than that in 1927. In the last eleven months of 1931 the tonnage per man-hour of transportation labor increased from 4.45 to 6.6.

Savings of a relatively minor sort, but nevertheless worth while, are frequently to be made by improving plant transport and haulage systems. Automatically controlled larries, such as used by the Hanna company at its Susquehanna mine, in Minnesota, for transferring ore from shaft to mill bin or stockpile, usually pay for themselves, by labor saving, early in the life of the equipment. Another example is the monorail tramway used at a Utah mine for transporting ore, concentrate, coal, and coke between mill and railway and in the return direction.

Electrification of open-pit work has usually begun at the shovels, the problem of carrying the current into and around the pit having at first been a deterrent to the electrification of haulage. This difficulty having been satisfactorily overcome, as evidenced in several pits in the Lake Superior Iron Country and elsewhere, the way would seem open for the complete electrifying of many pits now wholly or partly operated by steam. Partly electrified pits, having both electric and steam (or other power) shovels in service in conjunction with steam haulage, are numerous. At the Dundas quarry of the Canada Crushed Stone Company, in Ontario, methods have been steadily improved since 1905, since when production per

Table I—Comparison of Steam and Electric Haulage Costs, Utah Copper Company

	Direct Haulage Costs	
	In Ore per Ton	In Waste per Cubic Yard
Steam haulage — 1927	4.55c.	8.25c.
Average electric haulage — 1930-31	2.70	6.07
Indicated saving	1.85c.	2.18c.
Per cent reduction	40.7	26.4

man employed has risen steadily from 15 to 45 tons per 10-hour day. The latest installation was an electric shovel, and plans are now completed for electrifying the pit haulage. At Nevada Consolidated's Ruth pit, steam shovels are gradually being replaced by electric units with 4-yd. dippers.

Savings effected with the electric shovel (350-ton class), as compared with the steam machine, have been marked, with respect to labor, power cost, repairs, and maintenance ("Electric Shovels," D. J. Shelton and D. Stoetzel, A.I.E.E., April, 1925). It has eliminated firemen and craners, coal handling, and water lines. Power cost has averaged 0.5c. per cubic yard as compared with 2.5c. fuel cost per cubic yard for the steam unit. Cost of repairs and maintenance, averaged over a four-year period, have been less than $5,000 per year for the electric (350-ton) machine, as against $25,000 per year for the equivalent steam shovel. These savings represent a handsome return on the capital investment required for replacing steam with electric shovels compared with the interest to be earned by leaving the money in a bank or invested in average securities.

Air transportation has come to figure largely in the affairs of many mining companies, remotely situated or engaged in prospecting or developing properties in areas difficult to reach by ordinary means. An outstanding example is that of Bulolo Gold Dredging, Ltd., which has during the last year and a half been equipping a property in New Guinea for dredging, and also providing a 3,000-hp. hydro-electric plant. The property is 45 miles in an airline from the coast, and 2,000 ft. above sea level, without road or waterway through the jungle between it and the sea. In all, 2,500 tons of machinery, equipment, and supplies had to be transported. From April 1 to Oct. 31, 1931, 964 tons was freighted in by airplane at an outside cost of 5c. per pound, and from Oct. 31 to Dec. 12, 472 tons at 3.5c. per pound. In the latter period, lower costs were obtained by virtue of better weather conditions, which permitted transportation at a rate of 337 tons per month, as against 138 tons in the earlier period. These costs include everything except depreciation. The saving here cannot readily be calculated, but evidently it is considerable. The alternative to air transportation was to build and maintain a road, considerably more than 45 miles in length, through a difficult, primitive, and rainy tropical wilderness, and to move the equipment over it. As one views the dredge shown under construction in a photograph, recently received, it is hard to believe that every pound of it has been air-transported, as C. A. Banks, managing director, attests.

Pumping, hoisting, and practically every operation not yet mentioned that is involved in mining, offers a chance for cutting costs. Automatic control of centrifugal pumping has resulted in important savings at two Pennsylvania coal mines, but is slow to find adoption outside the anthracite field—this for no good reason. At one of the mines referred to, automatic starting and stopping alone of the pump in service saved 500 kw.-hr. per day, which, with power at 1.5c. per kilowatt-hour, meant a saving of $7.50 a day, aside from the pump runner's wages, a variable amount. At the other mine, automatic control saved $3,600 in ten months.

Late in 1930, the Tennessee Copper Company adopted automatically controlled centrifugal pumping in its Burra Burra shaft, at Ducktown, Tenn., in place of manually controlled pumping. The economy thus effected is revealed in the accompanying data. Table II gives the total pumping duty in the two shafts of the Burra Burra mine for a 181-day period. Table III gives the total cost of pumping during the 181 days from January to June, inclusive, 1930, when manual control was used, compared with the cost during the corresponding period in 1931, when electric control was employed. The cost data also cover small gathering pumps, both of air- and electrically-driven types, as well as the main shaft pumps.

Cost per 1,000 gal. pumped against 1,282 ft.-head, including the items shown in Tables III and IV, was $0.313 in 1930 and $0.161 in 1931. With the new equipment in 1931, pumping against a 1,440-ft. head, the cost per 1,000 gal. was $0.100.

The automatic equipment is supervised by one man. Repairs on pumps, motors, and pipe lines are done by a shop crew of one to four men, as required.

Automatic and semi-automatic hoisting have advantages in respect to labor saving when the hoist can work under the conditions for which it is designed. The most recently installed automatic hoist is probably that at the Emma Nevada shaft of Consolidated Coppermines—a unit designed to handle 5,000 tons per 8-hour shift, the automatic features of which have thus far been tested only, intermittent operation from more than one level having prevented operation without an engineer. Its safety features are though to have netted a saving, the hoist being practically safe from the hazard of manual control.

Faster hoisting from shallow depths is likewise contributary to economy, hoists having been designed especially for this purpose. Where cage hoisting is employed and a greater net load is wanted without increasing the total load on the hoist, the end can often be attained by adopting cages of light metals, such as introduced in Germany by at least one manufacturer.

(To be continued)

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Table II—Pumping Duty for 181 Days, Burra Burra Mine

Shaft	Total Water	Static Head, Ft.	Type Pump
McPherson	4,217,476	450	Electric centrifugal
Burra Burra (strong acid water)	6,538,630	749	Electric centrifugal
Burra Burra	39,820,000	1,440	Automatic centrifugal
Totals	50,576,106	1,268—Head-weight average	

Table III—Cost of Pumping, Burra Burra Mine, 181-Day Period

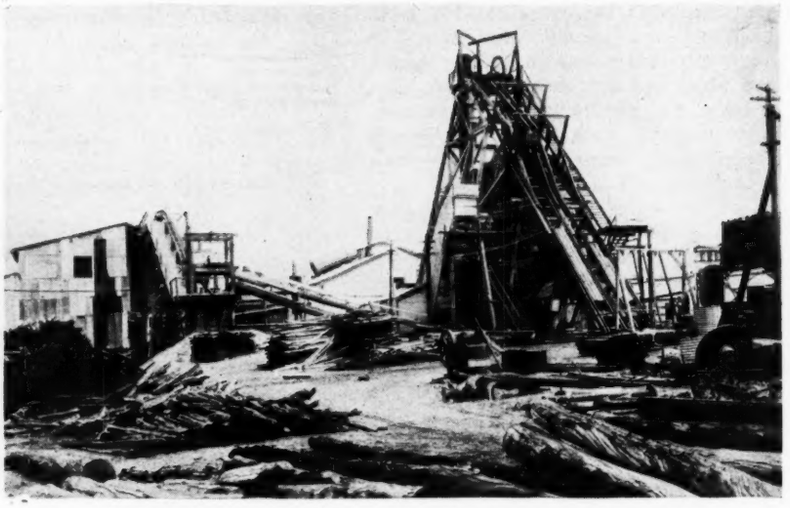
Period	Labor	Material	Electric Power	Compressed Air	Total
January-June inclusive, 1930. Manual control	$6,360.46	$4,678.83	$4,024.96	$612.29	$15,696.54
January-June inclusive, 1931. Automatic control	3,207.20	1,733.31	2,834.93	371.64	8,147.08

Table IV—Approximate Cost of Automatically Controlled Pumping on Water Thus Handled, Burra Burra Mine

Period	Labor	Material	Electric Power	Compressed Air	Total
January-June inclusive, 1931. 181 days	$1,200.00	$400.00	$2,346.10	.....	$3,946.10

Bunk house at Copper Mountain Mine, in British Columbia. An especially good fire-escape arrangement

Gold Metallurgy in Western Australia



Surface plant at the Sons of Gwalia mine, near Leonora

ALTHOUGH gold has been found in appreciable quantities at hundreds of small deposits throughout Western Australia, production has been principally from the famous Golden Mile at Kalgoorlie. The future of gold-mining in the state is unquestionably bound up with the success that the Kalgoorlie companies achieve in improving operating technique to make low-grade material payable. At increased depth, the Kalgoorlie orebodies are progressively lower in grade. They also lack the localized enrichments that were a feature of early operations in the district. The only hope for the companies obviously lay in the treatment of larger tonnages in modern plants. Until recently, production was controlled by a number of comparatively small mines (the largest of which treated only 130,000 tons in 1930) operating as separate units. This lack of united control did not make expansion easy.

Production of Western Australia declined steadily until 1929, when it amounted to only 377,164 oz., compared with the record of 1,962,361 oz., established in 1903. In 1930, there was an increase. The mines of the state treated 645,344 long tons of ore, which yielded 416,361 oz. of gold, increases of 16,944 tons of ore and 39,197 oz. of gold, respectively. The average yield of ore mined was 55s. 3d.,[1] an increase of 5s. per ton over the yield in 1929. In 1931, the increase has continued. For the first ten months of the year, production was 387,487 oz., compared with 340,105 oz. in the same period of 1930. Western Australia may again become an important gold producer, its total yield to the end of June, 1931, having been 38,529,000 oz., or about £163,000,000.

The first step in the permanent improvement of the industry was the consolidation, in September, 1929, of the Lake View & Star properties with the adjoining Golden Horseshoe Estate. Lake View & Star holdings included the Lake View, Hannan's Star, Chaffers, and Ivanhoe mines. For several years prior to the merger, Golden Horseshoe was closed but had ore reserves of 481,000 long tons assaying 7.7 dwt. per ton. One of the reasons for consolidation was the fact that important orebodies on the Lake View & Star holdings could be worked, economically, only through the Golden Horseshoe shafts. Immediately upon consolidation, extensive underground exploratory and development work was started, with the dual object of proving additional ore and linking up the workings of the mines for cheap mining. The underground staff was increased and a skilled geologist appointed to control prospecting. At the 3,300 level of the Ivanhoe shaft a connection was made to the Golden Horsehoe shaft and thence to the Chaffers. Electric storage battery locomotives, with a hauling capacity of 150 tons, were placed in commission on this level. In addition, programs of research into the problem of ventilation and all phases of mining were undertaken.

Erection of a new treatment plant to cost £300,000, with a final capacity of 40,000 tons of ore monthly, was begun at the Chaffers mine. Construction has been steadily continued, and the management hopes that early in 1932 the plant will be in a position to treat 30,000 tons monthly. The flowsheet at present is shown in Fig. 1. The method of treatment is new to Kalgoorlie in that the ore is concentrated by flotation and the small tonnage of concentrates roasted, prior to cyanidation.

Power costs at the Kalgoorlie mines have always been high at 1.5d. per kilowatt-hour, using the wood-steam or producer-gas, gas-engine systems. Lake View & Star has erected, on the Ivanhoe lease, a crude-oil plant consisting of two 500-hp. Crossley Premier engines. An 1,100 hp. engine and alternator have been placed on order. The equipment is so arranged that, in the event of shortage of oil supply, the engines can be run on producer gas. Oil consumption is given as 0.37 lb. per brake horsepower-hour. Oil costs £10 per ton at Kalgoorlie; it is about £3 10s. landed at sea board. This power plant, according

Contributed by a competent and qualified Australian mining engineer who, because of professional connections, is obliged to remain anonymous.

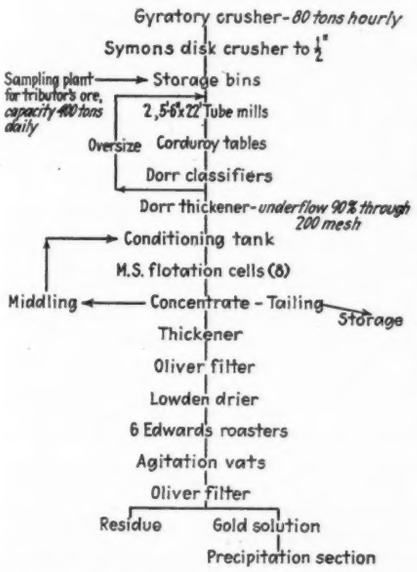
[1]Figures given in currency in this article are on the basis of the gold parity. That is, 84s. 11d. per ounce of gold.

Underground in a Kalgoorlie gold mine, with two air drills at the face

to recent estimates, will save £23,000 annually.

During the year ended June 30, 1931, Lake View & Star treated 173,111 tons for a return of £564,676, which is equivalent to 65s. per ton. These figures include high-grade ore shipped by tributors. When the larger tonnages are being treated working costs on company ore are not expected to exceed 20s. per ton. Treating the proposed 40,000 tons monthly, the increase in the state's annual output from Lake View & Star production will be at least 85,000 oz

At the time of its consolidation with Lake View & Star, Golden Horseshoe had on its leases a huge tailing dump—a prominent landmark at Kalgoorlie—estimated to contain 2,550,000 tons assaying 7s. 10d. per ton, of which 45 per cent was said to be recoverable. A company, Golden Horseshoe (New), Ltd., was formed to work this dump, and a



Flowsheet of the new Lake View & Star flotation unit, now operating

plant capable of treating 40,000 tons monthly, and costing £34,000, was erected. The company has treated about one-quarter of the dump at a cost of £50,000 and a profit of £40,000. The method of treatment is to sluice the tailings with hydraulic monitors, agitate with cyanide, filter in Oliver filters, and recover the gold by the Merrill-Crowe system. Total cost of treatment is 2s. per ton.

Further evidence that the higher level of gold production in Kalgoorlie will be maintained is found in the recent annoucement that Boulder Perseverance has decided to build a new treatment plant at a cost of £60,000. It will have a capacity of 250 tons daily. During 1930 this company treated 79,000 tons for a total return of £178,900, equivalent to 45s. per ton. This grade gave a mine profit of £14,857, and a total profit of £533, compared with a loss of £3,148 in 1929. The company is credited with 550,750 long tons of ore reserves assaying 10 dwt. With the new plant in operation, working costs may allow larger tonnages of junore to be mined.

For several years Boulder Perseverance has been experimenting with the bromo-cyanide method of ore treatment, by which the ore, after fine crushing, is agitated with cyanide solution, then bromo-cyanide solution, followed by filtering and precipitation as usual. Experimental work carried out by the company's metallurgist, Charles E. Blackett, has shown that by the use of bromo-cyanide the time of agitation can be reduced from sixteen hours as at present to three hours, and a recovery of over 90 per cent of the gold obtained. Mr. Blackett found that the following conditions gave the most satisfactory result:

1. Grind to minus 150 mesh.

2. Amalgamation to remove coarse gold that would not be dissolved in the time available for agitation. Tests showed that up to 20 per cent of the total gold may be removed by amalgamation.

3. Agitation of 1:1 pulp, for two hours, in 0.15 per cent potassium cyanide solution.

4. Addition of 1 lb. lime per ton of ore and agitation with 1 lb. bromo-cyanogen per ton of ore, for one hour.

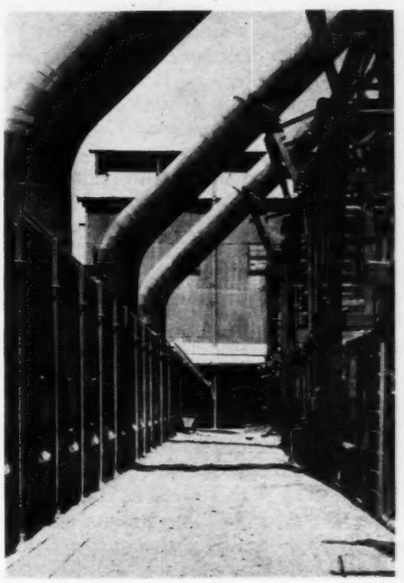
5. Addition of 2 lb. lime per ton to neutralize acidity and regenerate matallic cyanides.

The bromo-cyanide method, or Diehl process, is not new to Kalgoorlie, having been formerly applied to high sulphide ores and to telluride ores, to obviate roasting. The Boulder Perseverance ore contains 6.8 per cent FeS_2; 20.3 per cent $MgCO_3$ plus $CaCO_3$, and 63 per cent SiO_2.

Present-day cost of milling at Kalgoorlie in those plants in which the ore requires roasting is from 12s. to 15s. per ton. On the basis of 250 tons daily, the cost of the bromo-cyanide method is not expected to exceed 10s. per ton, of which fine grinding, agitation, and vacuum filtration will be the principal costs.

The proposed rate of treatment of 250 tons daily is only a little above the present output, so that the installation of the bromo-cyanide plant cannot be expected to make any appreciable increase in the gold production immediately.

The most important new Western Australian gold mine is at Wiluna, 120 miles east of Meekatharra. After an expenditure of £1,500,000 on underground development and the erection of a mill to treat 40,000 tons monthly, operations were started in April, 1931. Underground work has been laid out following most modern practice. Two electric hoists are in use, one with a capacity of 2,500 tons of ore daily from a depth



Flues for gases from the roasting unit in the Boulder Perseverance mill

of 2,000 ft. The second, operating in the man-material shaft compartment, hoists 45 men per trip. Five-ton electric battery locomotives have been installed along the main haulage levels—290 ft. and 450 ft. Electric pumps, automatically operated, are in use. The Wiluna treatment, in brief, consists of:

1. Crushing in gyratory and Symons crushers.

2. Fine grinding in Hardinge ball and Ruwolt tube mills.

3. Flotation.

4. Filtration of flotation concentrate in Oliver filters.

5. Drying concentrate in Lowden dryers.

6. Roasting in Edwards furnace with recovery of arsenic trioxide.

7. Cyanidation and gold recovery.

Wilfley pumps have been adopted throughout the plant. The power and compressed-air station consists of seven, 600-hp. National crude-oil engines for electric power generation, and three, 500-hp. Crossley Premier engines, each direct coupled to a Bellis & Morcom air compressor designed to give 2,000 cu.ft. at 90-lb. pressure.

Operations at Wiluna have, so far, not been entirely satisfactory. Underground ore dilution, with the sublevel system of stoping, has been greater than expected, and the grade of run-of-mine ore lower—30s. instead of 39s. The fine-grinding section of the mill is much under capacity and minor troubles have been experienced in flotation. Even with ore of considerably lower grade than the estimate a profit is expected. Working and development costs now total 23/- per ton. Existing plans contemplate restricting production to 30,000 tons monthly, and on this new basis the gold production may be about 100,000 oz. annually.

Outside of Kalgoorlie and Wiluna, the only large producer is the Sons of Gwalia, at Leonora. This mine, managed by Herbert Hoover 32 years ago, has produced 50 tons of gold, paid £1,128,000 in dividends, and is operating at a small profit. A few years ago, the Western Australian Government loaned the company £78,000 for development work in the bottom levels and for equipment. The main shaft has reached a depth of 4,000 ft. Development has shown that the continuity, width, and value of the orebody are being maintained. The company has increased its mill tonnage to 12,800 tons monthly and is continuing the treatment of tailings valued at 8/6 per ton. Should development work continue satisfactorily, the mill tonnage will be further increased. The method of treatment at the Sons of Gwalia is much simpler than at Kalgoorlie or Wiluna, and comprises:

1. Crushing in gyratory breaker.
2. Fine crushing in 30-head stamp mill.
3. Fine grinding in tube mills.
4. Agitation with cyanide and gold recovery.

The cost of mining and milling is 22s. 6d. per ton.

Intensive prospecting is being carried on in all parts of Western Australia, which, with its wide areas of Pre-Cambrian rocks, gives the prospector ample scope. Many options over new and old mines have been taken. Mergers of Kalgoorlie mines are again being discussed. Large prospecting concessions in the north-west part of the State have been granted by the State government, which, fully realizing all that the revival in the industry means, is cooperation with operators to the full. These are the obvious signs of renewed interest in gold mining. With an increase of about 190,000 oz. from Wiluna and Lake View & Star, Western Australia's gold production for 1932 should not be less than 600,000 oz.

Ore Reserves of Western Australian Mines

Mine	Tonnage (Long Tons)	Grade (Dwt. per Ton)
Lake View & Star...	1,431,453	8.3
Wiluna Gold.....	780,000	7.9
Great Boulder.....	80,372	9.8
South Kalgurli:		
Proved.....	50,000	9.8
Probable.....	288,000	8.5
North Kalgurli:		
Proved.....	20,000	20
	90,000	11
	15,000	8
Probable.....	75,000	8.7
	10,000	8

Based on most recent estimate published.

■ ■ ■

Electrical Trends and Developments

UNTIL recently, the larger and more important hoists in the metal mining industry were of the direct-current type using a variable-voltage system of control and with the motor-generator set arranged with a flywheel and automatic slip regulator to equalize the demands on the alternating-current lines, according to J. E. Borland, mining engineer with the Westinghouse company. With the extension in capacity of power systems, the necessity of smoothing out the peaks in demand of the hoisting cycle no longer exists in many installations, and the tendency is toward a more frequent application of the synchronous motor generator with hoist equipment. The latest installation of this type is the equipment recently ordered by Noranda Mines for its Noranda, Que., operation.

With the variable-voltage system and synchronous motor-generator set the peak loads are carried back to the power line and are increased by the losses in the motor generator, but the peak demand is not reached until the hoist has been brought to full speed and the demand comes on more gradually than in the operation of an induction-motor driven hoist. As compared to a flywheel motor-generator set, the synchronous set will usually show a slightly lower energy consumption.

Marked evidence of a wider application of mechanical loading in metal mining is available. This applies particularly to the great increase in the use of slusher or scraper hoists, largely in iron mining, and the development of larger and more powerful hoists of this type.

In the field of mine ventilation, fans and blowers are being applied in larger capacities and for higher pressures in order to ventilate the deeper workings of metal mines properly. With increased power requirements the need of better efficiency in fans and driving motors is of greater importance.

Electric power shovels of the latest mining type are being developed in larger sizes and with more powerful equipment, particularly with respect to the short-coupled type of mining shovel or modification of it, which has recently been extended to the five-yard size.

The variable-voltage system has been established as the most economical and flexible form of drive for electric power shovels in all but the small sizes. The present trend is toward the use of higher-speed motor-generator sets, enabling the use of more powerful equipment and a better mechanical arrangement of sets within the space limitations of the shovel cab.

Diesel-electric power is being used more extensively in such units as the large dragline excavators and in the smaller mining-type shovels. Up to now, the variable-voltage system has not been employed with such equipment, but as satisfactory high-speed engines of suitable dimensions develop, one may look for the extension of this form of drive to the type of shovel used in open-pit mining and quarrying.

Development of synchronous motors with high starting torques has resulted in the adoption of simplex synchronous motors for driving ball mills and the like, thereby to obtain the power-factor correction inherent in this type of drive. Such a motor has a phase-wound damper winding and starts as a wound-rotor induction motor. As it approaches synchronous speed, this winding is short-circuited and the usual d.c. field winding is excited. Thus the motor operates at full speed as a synchronous motor, but has the starting characteristics of a wound-rotor induction motor. An installation of this type was made recently by American Smelting & Refining.

Use of synchronous motors in the past has been limited to fairly clean working places. The recent purchase by the Anglo-Chilean Consolidated Nitrate Corporation of a synchronous motor having certain features of design to drive a gyratory crusher and to correct the power factor of the plant points the way to a new type of construction. This motor was totally inclosed, with a separate inclosure for the slip rings. The size of the motor for this rating was maintained close to that of the open motor by following the design of the fan-cooled totally inclosed induction motors, which have fans on the inside of the inclosure for stirring up the air, and fans on the outside of the inclosure for blowing air over the entire motor.

With respect to arc furnaces, standard practice in the past has been to change the voltage on them by changing transformer taps through the use of circuit breakers. Two or three operating voltages have been brought out to circuit breakers, and the furnace operator has selected what he wanted. Recently, the tendency has been to abandon this practice and to use a motor-operated, no-load tap changer mounted inside of the transformer tanks. This eliminates the use of the tap-changing circuit breakers and the floor space required for them. Operation of the equipment is simple—the main-line circuit breaker is opened, the tap-changer motor energized until the proper transformer tap selection is made, and then the main-line circuit breaker is closed again. Such equipment has been installed on practically all of the transformers for steel furnaces shipped during the past year.

COST OF DRIFTING

Morris J. Elsing

DRIFTING ordinarily comprises drilling and blasting, mucking and tramming. A third cycle is sometimes necessary when the ground is such that timber is required. The cost of each of these operations will be given separate consideration.

If ground conditions permit, to blast every shift regardless of the depth of the round is customary, provided, however, that at least three feet of rock will be broken. Commonest practice is to operate one-man drills. The drill runner is assisted—in setting up, tearing down, and blasting—by a mucker from a near-by stope or drift.

The most economical method of drifting where great speed is not required is when the drill operator has a clean "set-up." This avoids the necessity of "mucking back," which is an added expense and a disagreeable task. The driller puts in as deep a round as possible during the shift. He may be able to get an advance of from 3 to 6 ft. from the round. If the ground is so hard that less than 3 ft. can be broken, a shift and a half, or more, may be employed for drilling. Drilling can be most economically done if the driller has more than one drift in which to work. With one drift only he will usually lose considerable time.

In most mines, blasting in drifts is allowed only at the end of the shift, because of the effect on mine ventilation; but under some conditions it is allowed during lunch hour. When this is done, the timing of the ending of the drilling must fit in with the blasting time. To do this, perhaps the last holes have to be shortened, so that the round will be finished in the required time. This is undesirable, and it is also wasteful if the round is finished appreciably before blasting time. With two or three drifts in which he can work, the driller may be able to complete his round and tear down and set up in another drift; and where the inducement of bonus or contract is sufficiently great, he will drill several holes in the new "set-up." Thus, he may be able to blast three rounds in two days or four rounds in three days. This method is equally effective in drifting where the ground is so hard that a round cannot be drilled in one shift. To complete the round may require only one hour more of drilling on the succeeding shift, after which the "set-up" can be made in another drift.

The same conditions apply to mucking. To consider mucking of an average drift as a shift's work has long been customary. More time may be required to complete this work; but under favorable conditions the clean-up can be made earlier in the shift and mucking started in another drift.

Time studies of drifting show that in an eight-hour shift only from four to five hours are spent in drilling. The operator requires about one hour to set up his drill and about the same time to tear down and blast. The rest of the eight-hour shift is consumed in getting to and from his working place and at lunch. Other time studies indicate that of the four or five hours' drilling time, only 40 to 90 minutes is devoted to actual, effective drilling. The rest of the time is consumed in changing drill steel, tightening bar or column nuts, and changing drilling positions of the machine.

The number of holes drilled per round varies according to the hardness and breaking characteristics of the ground. Dividing ground into three classes as soft, medium, and hard from a tabulation of different mines of the number of holes necessary to break a round in a 5x7 ft. drift, indicates that in a soft drift from 6 to 11 holes are necessary; in a medium hard drift, from 12 to 17 holes; and in a hard drift, from 18 to 24 holes. The average footage broken in the soft drifts is from 4.5 to 6.5 ft. per round. In medium-hard drifts with blasting after one shift of drilling, the advance is from 3 to 4.5 ft. per round. For very hard ground, drilling requires from 1½ to 3 shifts and no general average can be established for the advance per round.

Table I shows the footage drilled per machine shift in 5x7 ft. drifts. No distinction is made between the light and heavy drifting drills.

Table II shows the powder consumption per foot of drift at various mines. These mines are representative of different parts of the country and of different rock formations. The average for 5x7 ft. drifts in medium-hard ground is about 10 lb. of powder per foot of advance.

Mining literature contains extensive data on the cost per foot of drifting. Many of these costs are direct costs

Table I—Footage per Machine Shift

Location	Reference	Feet Drilled per Shift	Remarks
Southeast, Missouri	I. C., 6170	80–100	Fair average, splendid conditions, limestone and shale
Mascot, Tenn.	A.I.M.E., Vol. 72	53.99	All development work, dolomitic limestone
Engels, Calif.	I. C., 6260	58.76	Year 1928, 11.3 holes, 5.2 ft. deep, 14-16 holes per round, firm diorite
United Verde, Jerome, Ariz.	A.I.M.E., Vol. 66	44.0	Porphyry. 66 ft. per round
		37.0	Schist, 88 ft. per round
		9.2	Massive sulphides, 80 ft. per round
		24.0	Massive sulphide stopes
		48.0	Schist stopes
United Verde, Jerome, Ariz.	*M. C. J.*, 4/30	36–48	Hard sulphide in stopes
		108.0	Schist in stopes
Hard Specular Hematite	I. C., 6138	25–30	Hard ore and waste, slate, quartzite, jasper, and siderite
Arizona Copper Mines		60–90	Average ground

Note: I. C. refers to U. S. Bureau of Mines Information Circular.
A.I.M.E. to Transactions of the American Institute of Mining and Metallurgical Engineers; *M. C. J.* to *Mining Congress Journal*.

Table II—Powder Consumption

Location	Reference	Pounds per Foot	Remarks
Old Dominion Globe, Ariz.	I. C., 6237	9.1	Average level drift, 5x7 ft. diabase
Morning Mine, Mullan, Idaho	I. C., 6238	11.3	Drift 6x8 ft., hard swelling ground; quartzite
Mogollon, N. M.	A.I.M.E., Vol. 72	10.6	Andesite and rhyolite; 1922
Silver King Coalition	A.I.M.E., Vol. 72	11.0	Drift 5x8 ft., limestone, quartzite, sandstone and shale
Southeast Missouri	I. C., 6170	13.5	Drift 6x8 ft. limestone, dolomite, and shale
Hard Specular Hematite, Marquette	I. C., 6138	12.5	Drift 8x8 ft. hard ore and waste
Hanover Bessemer, Fierro, N. M.	I. C., 6361	9.9	Drift 8x7 ft., limestone
United Verde Jerome, Ariz.	I. C. 6440	13.5	Drift 6x8 ft., schist, quartz prophyry, massive sulphide
Pecos Mine, New Mexico	I. C., 6368	9.4	Drift 5x7 ft., schist
Ducktown, Tenn.	I. C., 6149	11.2	Drift 8x8 ft., schist
Montreal, Montreal, Wis.	I. C., 6369	9.3	Subdrift 5x7 ft., iron ore
Spring Hill, Helena, Mont.	I. C., 6402	9.4	Drift 7x7 ft., hard and tough
Ray, Ray, Ariz.	I. C., 6167	5.3	Drift 6x8 ft., timbered
Morenci, Morenci, Ariz.	I. C., 6107	6.4	Supply drift, 6x8 ft.
Page, Page, Idaho	I. C., 6372	12.9	Crosscut 5x7 ft., quartzite
Black Rock, Butte, Mont.	I. C., 6370	12.5	Drift 7x9 ft., timbered
U. V. X., Jerome, Ariz.	I. C., 6250	6.1	Drift 7x9 ft., schist
Average		10.2	

only and do not include hoisting, waste disposal, insurance, supervision, and miscellaneous charges. An attempt has been made to present herewith some of the most complete costs, selected from published data, together with my own drift-cost records.

United Verde, Jerome, Ariz.

	Cost per Foot
Labor	$7.87
Explosives	2.44
Timber	0.41
Compressed air	0.55
Repairs	1.44
Supplies	1.38
Total	$13.09

Note: Reference *M. C. J.* September, 1927. Average cost of drifting 1926. Drifts, 6x8 ft. Holes, 9-28; average, 16.

Alaska Gastineau Mining Company, Juneau, Alaska

	Cost per Foot of Drifting		
	Slate	Gabbro	Schist
Labor	$4.87	$4.76	$4.86
Power	0.29	0.36	0.29
Supt. and general expense	2.05	2.39	2.20
Store expense	0.08	0.08	0.27
Tramming labor	2.64	2.86	3.35
Tramming	0.03	0.13	.....
Machine drill expense	0.89	1.10	1.08
Explosives	2.50	3.26	2.82
Lighting, carbide	0.07	0.07	0.09
Timber	0.12	0.10	0.54
Other stores	0.01	0.01	0.09
Totals	$13.55	$15.12	$15.59

Note: Reference, A.I.M.E. Vol. 63. Drifts, 7x8.5 ft., ditch for water; 1920.

Tri-State District, Missouri—Mine No. 1

	Cost per Foot of Drifting
Drilling labor	$2.95
Mucking labor	4.42
Explosives	2.38
Tramming, power	0.97
Labor insurance	0.48
Compressor	0.62
Track and pipe	0.46
Ventilation	0.43
Steel consumption	0.06
Blacksmith	0.70
Miscel., carbide and repair	0.66
Total	$14.13

Note: Reference, *E. & M. J.* May 18, 1929. Drift 7x7 ft.; limestone, sandstone, shale; 14c. more for hoisting.

Park Utah Mine, Park City, Utah

	Cost per Foot
Drilling labor	$5.97
Mucking labor	8.96
Explosives	2.44
Tramming, motor haulage and trolley installation	1.16
Liability insurance	0.47
Compressed air	0.64
Track and piping (installed)	1.48
Ventilating	0.57
Steel	0.09
Steel sharpening	0.48
Machines (repairs and depreciation)	0.80
Timber (about ⅓ timbered)	0.28
Miscellaneous (overhead and supplies)	0.50
Total	$23.84

Note: Reference, I. C. 6290. Size of Drift 8x8 ft. Medium hard rock, limestone, quartzite, quartz diorite. This is a haulage drift, and the extra items account for the higher cost than for an average 5x7 ft. drift.

In 1922 I had the supervision of a drifting contract at the Boras Leasing Company in the Warren mining district, Arizona. About 500 ft. of work was done per month. Two drifts were run, one on the 900 level and another on the 1,100 level. No other work was being done, and therefore no arbitrary segregation had to be made which would vitiate the results. The shaft was of two compartments and operated by a Denver Engineering Company electric hoist. An Imperial Type 10 compressor was used. Power was purchased from the Bisbee Improvement Company at about 2.75c. per kilowatt-hour. Insurance amounted to about 5.50 per cent of the payroll. As the contract was being run for the Phelps Dodge Corporation, about half of the supplies were purchased from the company at a reduced rate.

The costs per foot, which varied slightly from month to month, are herewith presented; they represent over 6,000 ft. of work; the formation was a hard, firm unaltered limestone. Labor, $7.02; insurance, $0.49; auto upkeep, $0.29; power, $1.52; repairs, $0.26; water, $0.05; office, $0.15; supplies and explosives, $3.78; total, $13.56.

Herewith are the costs of drifting at the Bisbee Queen property, in the Warren or Bisbee mining district, Arizona. This work was done in 1927 under my supervision. The shaft was sunk 823 ft. and about 6,000 ft. of drifting was done in medium hard limestone. No timber was used, and the shaft was dry. Power was purchased at an average of 2c. per kilowatt-hour. Average cost per foot of five months' work is as follows: Direct labor, $4.49; indirect labor, $2.27; power, $0.99; explosives, $2.83; mine supplies, $1.70; miscellaneous, $0.70; total, $12.98.

Direct labor includes all labor of drilling, mucking, and tramming. Indirect labor includes hoisting, caging, top landing, and supervision. Insurance amounting to 5.5 per cent was charged to labor. Power includes the total power used in hoisting, compressing air, operating blowers, and lighting. Explosives include powder, fuse, and caps. Powder cost $16.50 per 100 lb. delivered to the mine. Mine supplies include rails, fishplates, track ties, turn sheets, ventilating pipe, tools, oil, grease, drill steel, drill parts, carbide, and all of the incidental supplies. Miscellaneous includes auto trucking, accounting, office expense, taxes, water, and incidentals.

Davis Daly, Butte, Mont.

	Cost per Foot
Total labor	$8.33
Explosives	2.37
Compressed air	0.89
Rock drill repairs	0.75
Steel consumption	0.22
Steel sharpening	0.20
Supplies	0.72
Total	$13.48

Note: Reference A.I.M.E. Vol. 66. Powder 20c. per pound. Does not include hoisting or waste disposal. For driving 602 ft. of drift.

Mineville District New York

	Cost per Foot
Drilling and blasting	$7.66
Mucking and tramming	4.60
Track, air, electric, power, and water lines	1.80
Miscellaneous	1.40
Total	$15.46

Note: Reference *E. & M. J.*, February 2, 1929. Drift 10 ft. high by 15 ft. wide. Two heavy Leyners, 5 ft. per round. Drilled in 5½ hours. Mechanical loaders cleans round in three hours, 50 tons. Gneissoid beds and limestone. Ore medium hard.

Cost of Drifting at Bunker Hill, Copper Creek District, Arizona

	Units Used	Price	Total Cost	Units per Ft.	Cost per Ft.	
Labor	.....	....	$1,942.96	....	$6.48	$6.48
Explosives						
40% gelatine	2,100 lb.	18c.	378.00	7.0	1.26	
Fuse	5,600 ft.	¼c.	42.00	19.0	0.14	
Caps No. 6	1,000	1.9c.	19.00	3.3	0.06	1.46
Fuel						
Distillate	700 g.	22c.	154.00	2.3	0.51	
27 plus fuel oil	1,080 g.	8c.	86.40	3.6	0.29	
Gasoline	93 g.	24½c.	23.00	0.3	0.08	
Carbide	300 lb.	7½c.	22.50	1.0	0.08	
Smithy coal	800 lb.	1¾c.	14.00	2.7	0.05	1.01
Lubricants						
No. 7	67 g.	58c.	38.86	0.02	.....	
Cup grease	30 lb.	12½c.	3.75	0.01	.....	
Machine oil	.....	....	5.00	....	0.16	0.16
Steel — ¾ in. hollow, box	136 lb.	23c.	33.35	0.11	0.11	
Drill repair parts	.....	....	15.00	....	0.05	0.16
Truck	1,176 miles	10c.	117.60	....	0.39	0.39
Assaying and maps	.....	....	73.80	....	0.25	
Supervision, sampling	.....	....	334.11	....	1.11	1.36
Accident insurance ($1,942.96 @ $7.79)	.....	....	151.36	....	0.50	0.50
					$11.52	$11.52

Costs are for 300 ft. of drifting done on the 300-ft. level under contract at $9 per foot, contractors furnishing all labor, explosives, lubricant, drill repair parts, and fuel except smithy coal, and ore sorting, which required an extra man about half time. Contractor also furnished truck.

Prices are at Winkelman, Ariz., 35 miles by road. Contractors hauled their supplies.

No track material or pipe included, as this was reclaimed from other parts of the mine.

Five-sixths of ground fair to good drilling and breaking; ⅙ hard, tight diorite.

Mine averaged 36 buckets water per day; about 2,600 gal.

Average wage about $4.60. Average feet per drill shift, 3.53; per round, 3.12 ft.

Time: July 10 to Sept. 15, 1926.

Hoist, 15 hp. with 10-cu.ft. bucket. Engine, 25 hp., belted to 9x8 compressor.

This work was done under the supervision of Marc Bailey.

No other work was going on, and this cost is therefore accurate for driving drifts in fairly hard limestone. About 1,000 ft. of work was done per month.

One of the most complete costs of drifting ever published is that of the Tintic Standard, at Eureka, Utah. This statement appeared in *Mining Congress Journal* of September, 1928. The drift was driven for the North Lily Mining Company, and settlement for the work was made on the basis of this outline of cost items. A mechanical mucker was used, and the drift was driven in limestone. Because of the detailed completeness of this cost, it is given in full.

To estimate the cost per foot of drifting within rather close limits is entirely possible. Under normal working conditions the rock hardness has no important bearing on the cost per foot except in extremely hard or soft rock. This is largely because the fixed costs of drifting comprise more than 50 per cent of the total cost. Assume, for example, a 5x7-ft. drift is to be driven.

Table III—Costs for Driving a 5x7-Ft. Drift

	Cost per Foot		
	Soft	Medium	Hard
Fixed Supplies			
Air pipe	$0.30	$0.30	$0.30
Water pipe	0.12	0.12	0.12
Rails	0.56	0.56	0.56
Spikes	0.04	0.04	0.04
Track ties	0.16	0.16	0.16
Ventilating pipe	0.62	0.62	0.62
Miscellaneous	0.40	0.40	0.40
Fixed supplies, total	$2.20	$2.20	$2.20
Fixed Labor			
Caging, top landing, placing track and pipe, hoisting	$2.60	$2.60	$2.60
Mucking and tramming	1.75	1.75	1.75
Power	.75	.75	.75
Insurance	.35	.35	.35
Fixed labor, total	$5.45	$5.45	$5.45
Total fixed costs	7.65	7.65	7.65
Variable Costs			
Drilling	$1.12	$2.25	$3.37
Sharpening steel	0.35	0.70	1.05
Drill repairs	0.25	0.50	0.75
Explosives	1.13	2.25	3.38
Variable costs	$2.85	$5.70	$8.55
Total cost	(a) $10.50	(b) $13.35	(c) $16.20

(*a*) For ground which requires from 6 to 11 holes, often drilled with a jackhammer held in the hands or by improvised standard. It applies to comparatively soft decomposed porphyry and schist, and also some auger ground.

(*b*) For ground which requires 15 or 16 holes drilled with either light or heavy mounted drifting drills. The ground is medium hard limestone, quartzite, shale, and the igneous rocks. It is more or less the average ground encountered in drifting in unaltered formations.

(*c*) For ground which requires more than 18 holes. The ground is hard metamorphosed and silicified limestone, quartzite, and igneous rock. The cost, representative of drifting in garnet limestone does not apply to the very hard and tough ground known as massive siliceous sulphides, silica, and pyrite, or some of the intensely silicified igneous rocks.

Cost of Drifting at Tintic Standard, Eureka, Utah

December, 1926—Advance, 625 Ft.

			Cost per Foot
Direct Charges			
Labor	$3,922.15	.....	$6.28
Explosives	1,393.16	$2.23	
Rails and fittings	259.18	0.41	
2-in. air line	259.62	0.41	
Ventilation pipe	444.19	0.71	
¾-in. air line	78.19	0.13	
General supplies	58.10	0.09	
Timber	76.62	0.13	
Totals	$6,491.21	$4.11	
			4.11
			$10.39
Indirect Labor			
Heat, water, light	$36.70		
Drill sharpening	14.60		
Machine shop	45.45		
Superintendence	12.51		
General office	20.85		
Mine office	41.70		
General	43.37		
Team truck and auto	8.34		
Assaying 84 assays	63.00		
Hoisting men, 84 trips	35.28		
Engineering, 625 ft.	312.50		
Blasting caps, 1,823	21.88		
Compressor, 28 days	42.00		
	$698.18		1.12
Indirect Supplies			
Carbide, 417 man-shifts	$8.34		
Team truck and auto, 417 man-shifts	10.01		
Heat, water, light, 417 man-shifts	27.11		
General 417 man-shifts	8.34		
Machine-drill oil, 28 days	4.20		
Drill-sharpener oil, 28 days	4.20		
Mucker machine oil, 28 days	16.80		
Compressor supplies, 28 days	7.00		
Ventilation fans, oil per month	1.00		
Car grease, per month	3.00		
Hoisting men, 84 trips	5.88		
Drill steel per month	10.00		
Engineering supplies, 625 ft.	100.00		
Office supplies, 625 ft.	125.00		
	$330.88		0.53
General			
Power compressor, 28 days	$140.000		
Mucking machine 28 days	70.00		
Ventilation, 28 days	70.00	$280.00	0.45
Mucker depreciation, 625 ft.	$312.50		
Mucker maintenance, 625 ft.	156.25		
Plant depreciation, 28 days	287.84		
State and county taxes, 28 days	22.68	$807.27	1.29
Totals	$8,607.54		$13.78

The approximate cost, per foot, of materials and supplies used in this drift are: Air pipe, $0.30; water pipe, $0.12; rails, $0.56; spikes, $0.04; track ties, $0.16; ventilating pipe, $0.62; total, $1.80.

The supplies have a certain salvage value after the drift has been run and abandoned; but cost of recovery and loss caused by damage would amount to about 50 per cent.

The charge indicated is perhaps too high, and costs should be credited with this salvage value; but for the purpose of this calculation the total will be included.

The labor cost of mucking per foot is estimated as $1.75. This does not vary because of hardness of formation. As a matter of fact, in a soft drift requiring timber, the 5x7-ft. cross-section necessitates the removal of more muck than a rock drift of equal cross-section. However, the variation is not sufficiently great to warrant an attempt to calculate it.

The indirect miscellaneous labor of caging, top landing, laying track, and putting in pipe depends entirely upon the advance made, and, therefore, is a fixed charge per foot of drift.

Power is another fixed charge nearly independent of the character of rock formation. The amount used in hoisting has little bearing on the type of formation in which drifting is being done. Hardness of the ground has a direct bearing on the amount of air used in drilling; but the compressor probably is operating for other work, so the small amount of air used for the extra drilling is not of sufficient importance to consider power as a variable factor in drilling. The cost of power

Table IV—Summary Cost of Drifting

(Not Including Amortization of Equipment)

Location	Reference	Cost per Foot	Remarks
Engels, Calif.	I. C., 6260	$13.09	Drift 6x7 ft.; hard diorite; total cost year 1929
Barr mine, Tri-State	I. C., 6169	10.00–12.00	Drift 7x7 ft. to 8x10 ft., untimbered; cut holes 8 ft.; others 7 ft.; contract price $6 labor and explosives; Total cost year 1929
Alaska Juneau	A.I.M.E., Vol. 72	11.65	Haulage drifts 9x9 ft. Year 1929
		11.25	Intermediate 5x7 ft.; gabbro and medium slates
		12.10	Crosscuts 7x7 ft.; 60 per cent of these costs are for contracting labor and explosives.
Mogollon, Dist., N. M.	A.I.M.E., Vol. 72	11.65	Drift 5x7 ft.; andesite and rhyolite; little timber necessary. Year 1925
Aztec Mine, Baldy, N. M.	A.I.M.E., Vol. 72	15.89	All development 1918-1921 including $7.89 for explosives, mucking, timbering, drilling. Sandstone shale, and porphyry
Utah Copper	A.I.M.E., Vol. 72	4.96	Drift 5.5x6.5 ft.; drilling and mucking, $2.25; explosives $0.68; air, $0.21; miscellaneous, $0.47 timbering, $1.35; Total, $4.96. Soft porphyry. Year, 1916
Miami Copper, Miami, Ariz.	A.I.M.E.	19.95	Haulage drift 10x10 ft. Year 1930
	TP314	10.70	Grizzly drift 5x7 ft., schist
United Verde, Jerome, Ariz.	A.I.M.E., Vol. 66	34.38	Drifts 6x8 ft.; sulphide. Year 1918
		16.31	Schist
		9.73	Porphyry
		21.22	All development year 1918
	M C J., Sept. '27	13.08	Average cost of drifting 1926
Iron Cap, Globe, Ariz.	A.I.M.E., Vol. 72	7.31	Drift 5x7 ft. Year 1925. Direct costs only
Davis Daly, Butte, Mont.	A.I.M.E., Vol. 66	13.48	Does not include hoisting or waste disposal. Year 1919
Tintic Standard, Eureka, Utah	*M.C.J.* Sept. '28	13.78	Driven for North Lily Mining. Year 1926. Mechanical mucking
Alaska Gastineau	A.I.M.E., Vol. 63	13.55	Drifts 7x8.5 ft.; slate
		15.12	Gabbro. Year 1920
		15.59	Schist
No. 1 Mine, Tri-State	I.C., 6113	14.13	Drift 7x7 ft.; limestone, sandstones, shales; 14c. more for hoisting. Year 1929
Mineville, N. Y.	I.C., 6092	15.46	Drift 10 ft. high x 15 ft. wide. Medium hard. Year 1929. Not hoisting
Park Utah, Utah	I.C.	23.84	Drift 8x8 ft. Year 1929. Main tunnel. Medium hard limestone, quartzite, diorite
Bisbee Queen, Bisbee, Ariz.		12.98	From personal experience. Year 1927. Only work going on was drifting. Limestone
Boras Lease, Bisbee, Ariz.		13.56	From personal experience. Only work drifting Medium hard limestone; 6,000 ft. of drifts
Beatson, Latouche, Alaska	A.I.M.E., Vol. 76	10.97	Slate, graywackes, schist
Pecos, N. M.	I.C., 6368	10.47	Granite, diorite, schist; 1929
		11.30	Year 1927-1929 inclusive
Hanover Bessemer, Fierro, N. M.	I.C., 6361	12.04	Granodiorite, haulage drift 8x7 ft., main adit. Year 1930
Eighty-Five, Lordsburg, N.M.	I.C., 6413	8.62	Hard andesite; direct costs only. Year 1930
Black Rock, Butte, Mont.	I.C. 6370	7.00	Contract price of labor for breaking, shoveling, tramming
Bunker Hill, Kellogg, Idaho	I.C., 6160	.....	Exploration work. Year 1928
Mine No. 8, S.E. Missouri	I.C. 6160	7.87	Direct costs; 7x8 ft. drift. Year 1928
Mine No. 2, Tri-State	I. C. 6121	9.00	Contract price labor and power, 7x7 ft. drift. yr. 1928
Mine No. 2, Tri-State	I. C., 6121	9.00	Drift 7x7 ft. Contract price labor and powder. Year 1928
Copper Creek, Arizona		11.52	Prospect work 35 miles from railroad. Information from correspondence
Verde Central, Jerome, Ariz.	I. C., 6464	11.98	13,052 ft. of drifting year 1927. Hard, firm greenstone and quartz porphyries

has a much more important effect on the cost of breaking ground than the character of the rock. Liability insurance is estimated at $0.35 per foot, and miscellaneous supplies estimated at $0.40 are not appreciably affected by the hardness of the ground.

Hardness of ground has its most important effect on drilling labor, sharpening steel, drill repairs, and explosives. In the average medium-hard drift these are estimated as follows, per foot: Drilling, $2.25; sharpening steel, $0.70; drill repairs, $0.50; explosives, $2.25; total, $5.70.

In a soft drift these costs have been assumed to be 50 per cent lower and in a hard drift 50 per cent higher. Table III shows the estimated costs for driving a 5x7-ft. drift in soft, medium, and hard ground under normal conditions where the muck is being hoisted and track and pipe are left in the drift. Costs for extremely hard ground or soft spiling ground are not considered.

The estimated cost of timber and timbering in a 5x7-ft. drift is shown in the following calculations:

Timber 10x10 In. 5 Ft. C. to C.	Bd. Ft.
Posts and cap	171
Lagging	125
Spreaders, blocks, wedges	36
	332
	Per Set
332 bd.ft. @ 4c	$13.28
Per foot of drift	2.65
	Per Foot
Labor of timbering per foot of drift	1.15
Total	$3.80

Adding this cost per foot for timber and timbering to the estimated cost of drifting for soft ground results in a total cost of $14.30 per foot for a timbered drift in soft ground, which is approximately $1 more than the per-foot cost of a medium-hard drift.

The average of the most complete of 14 drift costs shown in the summary given in Table IV is $12.87, which corresponds closely to the estimated cost of $13.35. Fixed cost of labor and the fixed supplies is shown to comprise more than 50 per cent of the total cost of drifting. These fixed costs do not vary greatly in different parts of the country, and my opinion is that a fair average cost of driving a 5x7-ft. drift, untimbered and in medium-hard rock, will be between $13 and $14 per foot; and that the variations from this price for drifts shown in the table are due to different working conditions and ground conditions, different methods of bookkeeping, and varying efficiency of management. In other words, the cost of drifting is closely similar all over the United States, and the wage scale has little or no bearing on the ultimate cost of the work. High efficiency of management is by far the most important factor in obtaining low drifting costs. This can be shown in a multitude of ways, as the correct style and size of drilling machine, properly tempered drill steel, best style of drill round to break the ground, the most effective strength of powder and caps, tamping, number of working faces, plenty of cars, good shovels, and other related factors.

■

Asbestos Mining in Quebec

Fig. 1—King pit of Asbestos Corporation in left foreground; Beaver mine, left background. Right foreground shows overburden, surmounted by cableways. King mill to right.

W. A. RuKeyser

IN *Engineering and Mining Journal-Press* of April 15 and 22, 1922, I contributed a series of two articles on "Asbestos Mining and Milling in Quebec." The economic importance of this mineral in time of peace as well as war, and the development of deposits in Rhodesia and Russia in the last decade, suggest the publication of a series of articles, of which this is the first, describing technical methods. Of interest is the fact that action has been taken in regard to recommendations made in 1922, to the effect:

1. That if the Canadian industry was to compete successfully with increasing production elsewhere, a consolidation was necessary of many small and comparatively inefficient units, with the elimination of marginal procedure. This was necessary from a business as well as a technical point of view. In the years following the War, cut-throat sales policies of competing companies threatened the extinction of the Quebec industry. The technical handicaps to efficient mining were yearly becoming more and more accentuated by the depth of the open-cut operations, coupled with the physical limitations imposed by the property lines of the various companies.

2. That an extension of the use of power-shovel methods, as contrasted with hand-loading and cableway hoisting, was desirable.

3. That piston drills should be displaced by deep-hole blasting, using high-speed, mounted drills.

4. That a technically trained operating personnel was necessary, with experience on other types of deposits, and metallic mining methods.

5. That necessity for proper exploration of the orebodies should be realized, and reserves blocked out and delineated.

6. That cyclone crushers in the mills should be replaced by jumbos.

7. That the number of grades of fiber produced by competing companies should be reduced.

8. That flowsheets should be simplified and mechanical mill details modernized.

Eighteen independent producing companies were operating about 24 mines and mills in 1920. In 1930 the number had been reduced, by consolidation and shutting down of marginal producers, to seven companies, operating ten properties. In 1926 Asbestos Corporation, Ltd., was formed, to succeed Asbestos Corporation of Canada, Ltd. This consolidation brought under one control the King mine and mill; the Beaver, British-Canadian, Fraser, the Vimy Ridge, and Thetford properties of the Bennett-Martin company; the Consolidated group; the Federal, Belmina, Kitchener, Maple Leaf, Asbestos Mines, Ltd.; the Pennington and the properties of the Black Lake Asbestos & Chrome Company. A few other properties that had never proved profitable were also included, so that almost the entire known orebody was in the hands of Asbestos Corporation and six other companies.

Asbestos Corporation then proceeded to centralize and modernize its operations. The King mill, destroyed by fire, was replaced by a steel and brick building, and important changes were made in the flowsheet. The old Beaver mill was replaced by one to serve the Beaver and Consolidated mines. Vimy Ridge was kept in operation, but revamped; also the British Canadian at Black Lake and the Maple Leaf. Plants at the other properties were either razed, the equipment salvaged, or, as in the case of the Federal at Robertsonville, held in reserve.

With the exception of Asbestos Corporation, with milling capacity of over 500 tons per hour, and the Johnson's Company, with milling facilities for 130 tons per hour, all remaining properties are under control of manufacturers of asbestos products. The Rhodesian output is mainly controlled by British manufacturing interests — Turner-Newell, Ltd.—so the competition is confined to a combination of two independents of Canada and the Soviet State Trust—Uralasbest. However, when Russia can sell asbestos in the American market cheaper than it can be produced by the manufacturing companies owning mines in Quebec, the Russian fiber will replace the Canadian fiber, at least for many purposes. Milling capacity of those properties that have their own product outlet is considerably in excess of present subnormal requirements. For that reason I am taking production and cost figures of 1928 as typical and representative.

Chrysotile asbestos occurs in one of two modes. The first, in which the cross-fiber veins are associated with serpentine derived from limestone or dolomite intruded by diabase, is illustrated by the asbestos of Arizona, Carolina (South Africa), and Minusinsk (Siberia). The second occurrence is in serpentine derived from peridotite and/or pyroxenite. This association, the more important, characterizes the Quebec, Rhodesian and Ural deposits. The geology of these deposits is generally not complex, although questions of origin are involved. In Quebec, the so-called serpentine belt, which gives

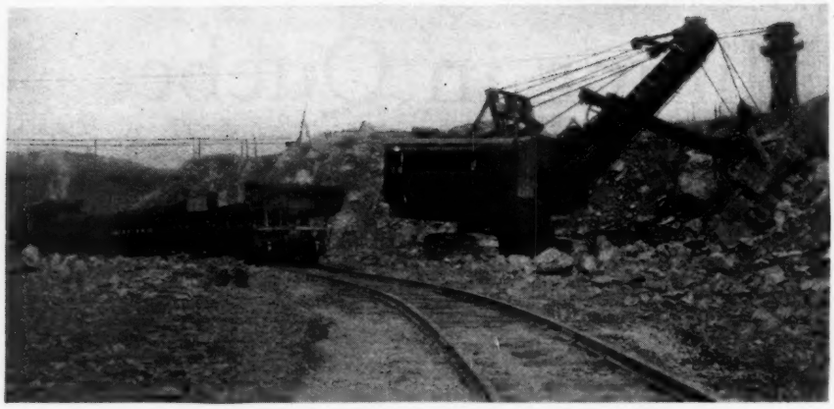


Fig. 2—Marion electric shovel at the Jeffrey mine

rise to the occurrence of the mineral, extends in a broken outcropping from northern Vermont almost to the Chaudiere River in Beauce County, Que. The main productive center is at Thetford Mines, midway between the United States border and Quebec City. The strike of the productive zone is nearly northeast-southwest. The main producing zone extends southwesterly to Black Lake, 5 miles from Thetford, and less important production has been obtained as far as Coleraine, 5 miles from Black Lake.

The next important producing locality on the belt is near Danville, the operations of the Johns-Manville company, 40 miles southwest of Thetford. Northeast from Thetford the productive zone extends 16 miles to East Broughton. The serpentinized peridotite-pyroxenite complex attains its maximum width of 15,000 ft. at Black Lake, and narrows to about 5,000 ft., as far as is known, at Thetford. The next outcropping, 3 miles northeast, exposes the serpentine as a narrow, inclined-sill structure with a maximum width less than 1,000 ft. Here is found the transition phase between cross-fiber and slip-fiber asbestos. Two miles away, at Robertsonville, the workable belt is from 300 to 500 ft. wide, the badly sheared and slipped serpentine dipping about 70 deg. southeast. Footwall and hanging-wall contacts are sharply defined, the former consisting of black slates and quartzites, the latter of gray quartzitic rocks, greenish and purple slate, and schist. On the slip-fiber deposits is no residuary peridotite, nor the intrusive acid afterphase—granite and aplite.

The asbestosized serpentine at Thetford occurs in productive "heads" or zones, between which are barren or nearly barren ones. Gradation between ore and non-commercial rock is seldom sharply defined. In the Quebec deposits, usually the longest and "silkiest" fiber is to be found closely adjacent to granitic or aplitic intrusions. Structurally, the largest of these is in the form of a dike, over 60 ft. wide. More usually the acid intrusive afterphase of the peridotite-pyroxenite-gabbro complex occur in the form of small fingers and stringers, irregularly distributed throughout the batholith. These acid rocks are extremely tough, hard and dense; they must be removed before milling.

Chrysotile asbestos is essentially a crystallized silicate of magnesia. The average analysis, according to eleven samples given by Cirkel, shows the silica at 40.49 per cent; magnesia at 41.41 per cent; and water of crystallization at 14.06 per cent. Lime and iron are impurities. Usually a little alumina is present. The chrysotile derived from limestone deposits is remarkably free from iron, whiter in color, and best adapted for electrical uses, but this occurrence may also result in a higher lime content, making the fiber more brittle. In the Quebec type of deposit the asbestos occurs in veins from a fraction of an inch up to a maximum of 3 in. wide. Veins and veinlets are distributed throughout the entire rock mass, but following definite zones. The veins are often roughly parallel to each other in two directions, forming an acute angle.

Chrysotile has virtually the same chemical composition as the serpentine that incloses it, so that property cannot be utilized in its extraction, nor can specific gravity, density, and the like. Two properties only have been so utilized. The first, which forms the basis of most milling operations to date, takes advantage of the fact that asbestos, because of its fibrous structure, can be "opened" by impact, and thus made amenable to separation from the gangue by air suction. The second, the basis of a process for which patents are pending, utilizes the friability of the gangue, as opposed to the high tensile strength of the asbestos fibers. Mining and milling of this mineral are further complicated by the extreme difference in values, dependent upon fiber length and quality. Thus, during the peak prices following the War, No. 1 Crude—lump material plus ¾ in. in unbroken fiber length, sorted out before milling—brought from $2,000 to $3,000 per ton; whereas the shortest grades, resulting from milling, brought $15 to $20 per ton. At today's prices, from $8–$10 to $250–$400, the range is not so extreme; but it is still sufficient to indicate the need of maintaining maximum possible fiber length during mining and milling—an outstanding peculiarity of asbestos technology. This fact also partly explains why the extraction of asbestos is perhaps more of an art than a science; and why flowsheets of contiguous mills, on the same deposit, will differ so radically. In no other branch of concentration practice does the design of flowsheet depend so much upon the dictates of the individual operator.

Only during the past decade—these deposits have been continuously mined since 1878—has a real attempt been made to delineate the orebodies by systematic drilling and other methods of prospecting. The mines had been started by open-pit methods from the grass-roots on as rich a showing of

Fig. 3—Movable head towers, cableway system, King mine, now used primarily for ore hoisting from shrinkare stopes

fiber as outcrops or casual surface prospecting opened up. The operators stuck to these showings, vertically and areally, so long as they persisted. When milling operations were commenced, the mills were sometimes built on what later proved to be the richest zones. Prior to 1920, when the industry had finally become extremely important, employing large capital, desultory diamond and Calyx drilling was done to supplement test pitting and trenching. In 1919 the first systematic drilling program in the district was initiated.

For several years past, extensive diamond-drill campaigns have been attempted. That of the Asbestos Corporation, at the King, Beaver and Bennett-Martin mines, is said to involve a total of over 100,000 ft., of which 57,000 ft. was done in 1929 alone. For perhaps the first time in the history of the camp, ore reserves are being definitely blocked out. Open workings having a maximum depth of nearly 400

ft. at the King Mine, underground work at the 500-ft. level, and drill cores from 1,700 ft. deep, have proved no variation in content or grade with depth. This was to be expected in view of the deep-seated batholithic structure of the parent cock, coupled with the fact that possibly 3,000 ft. had been eroded by glacial action.

Drilling for 4,200 ft., by means of a Calyx machine, was done in 1928 and 1929 at the Vimy Ridge mine of Asbestos Corporation for the purpose of prospecting and blocking ore. This type of drill proved eminently satisfactory. In my opinion the shot drill will do nearly as fast work on serpentine as the diamond drill, with lower costs per foot for holes from 60 deg. to vertical. The larger diameter of core produced is a great advantage in asbestos work, where chemical analysis for the determination of asbestos content is not possible. Results must be interpreted visibly or by actual mill tests of the cores.

Hydraulic stripping of overburden, attempted on a large scale at the King mine, has been abandoned. Boulders in the glacial drift caused trouble, as did also operations under the extreme winter conditions existing. At this property, where the maximum overburden is 120 ft., stripping is done by steam shovel; haulage by locomotive. This method is now adopted throughout the district wherever depth of overburden and areal extent warrant. Up to depths of 6–8 ft., hand methods are still used, with horse haulage. Thus, in 1928 and 1929, 600,000 cu.yd. was removed at the King; 100,000 at the Vimy; 50,000 at the Beaver; 10,000 each at the Maple Leaf and British Canadian—a total of 770,000 cu.yd. by the Asbestos Corporation alone. Prior to those years, development work had been neglected, both as to blocking out new ore as well as stripping overburden preparatory to future mining.

During the past five years the cableway system of mining, almost in general use at the Thetford and Black Lake mines, has been superseded by:

1. Crane and/or power-shovel loading, with locomotive haulage and power-hoisting up inclined planes or tunnels, as at the Beaver, the Johnson's Thetford property, and the Bell—the first to use this method in the district.

2. Shrinkage stoping, drawing through chutes directly into cableway boxes trammed by hand or electric locomotive through an adit to the floor of the open pit, where the boxes are hoisted by high-speed cableways. This method, initiated at the old Jacobs mine in 1917, now used at the King, is fast replacing high-bench mining in the open. But even at this property an electric shovel has been added to the pit equipment.

3. Power shovel and inclined-slope haulage (not tunnel), as at the Maple Leaf. Cableways therefore remain at only one major property—the King—and here they are gradually being abandoned, their primary purpose now being to hoist the ore coming from shrinkage stopes.

At Danville, where the sorting of crude—No. 1 is lump material, plus $\frac{3}{4}$ in. long; No. 2, plus $\frac{3}{8}$ in.—has never been a determining factor, the Johns-Manville company has for many years employed the spiral system of open-pit mining, using steam shovels and locomotive haulage direct to the mills. The steam shovels have recently been replaced by electric shovels. The Nicolet, a smaller property, also utilizes shovel methods, but with a shuttle-skip car that is hoisted directly to the mill bin. At East Broughton, on the slip-fiber deposits, steam shovels are still used. Early in the past decade, when consulting engineer for the Black Lake Asbestos & Chrome Company, I studied the economics involved in the sorting by hand of crudes in the mine before loading the skips or cars with waste and mill rock. At that time the value per ton of No. 1 crude was over $1,500. Necessity for sorting crude determined the mining methods used, so the cost of this crude was the difference between the cost of sorting and loading by hand and of power-shovel methods, involving fracture into pieces sufficiently small for men to handle conveniently—another added cost per ton of rock mined. Even with the high price of crude at that time the point was open to debate as to whether or not crudes were worth attention if over 2,000 tons of rock per day was to be mined. That particular property contained a total asbestos content of less than 2 per cent. In the district at large, with content over $5\frac{1}{4}$ per cent, the value of the crudes produced outweighed the added cost of mining. In the last five years, however, with (1) the application of the molded brake lining for automobiles, which has nearly eliminated the major market for crudes; (2) the rapid development of the Rhodesian and Russian fields; (3) the continually decreasing yield of crudes in the rock as mined; (4) the decreasing proportion of crudes yielded in proportion to tonnage of milled grades; and (5) the decrease in price of crude, the main economic consideration in favor of hand methods versus mechanical is disappearing. In my opinion power-shovel mining without previous sorting by hand will replace all other methods in open-cut mining to reasonable depths of asbestos in Canada. The introduction or extension of the use of indoor picking belts after the primary crushers will permit the sorting of sufficient crude to take care of market obligations. The rock thus sorted could be sent to the hand-cobbing sheds or to the mechanical process recently developed, whereby the cobbing, cleaning, and grading are done mechanically. If 30c. per ton of rock handled were thus saved by the use of large power-shovels, deep-hole blasting followed by crushers taking shovel-size rock, a conservative estimate of the saving over hand sorting and loading methods on a 2,000-ton per day basis would be $600. This means that at present prices, even if all the crude went to the mill proper, to be recovered as long-spinning fiber, the operation would show a net saving of $300 per day. For such reasons as these is due the ever-increasing use of power shovels in the Quebec district, with the complete avoidance of sorting crude in the pits made practicable in the near future.

Capt. J. G. Ross, mines manager for Asbestos Corporation, has kindly supplied the following information concerning the transition from open-pit mining

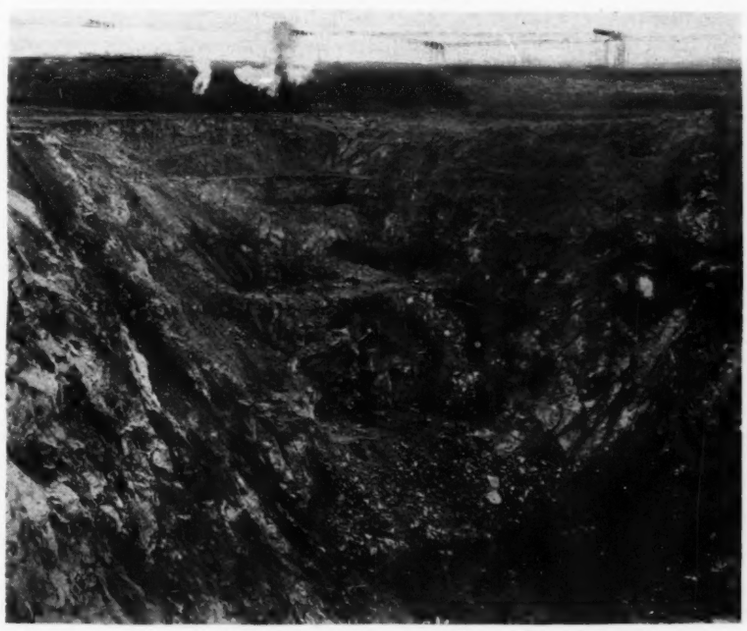


Fig. 4—King mine; four benches; total depth, 390 ft.; stripping overburden in background.

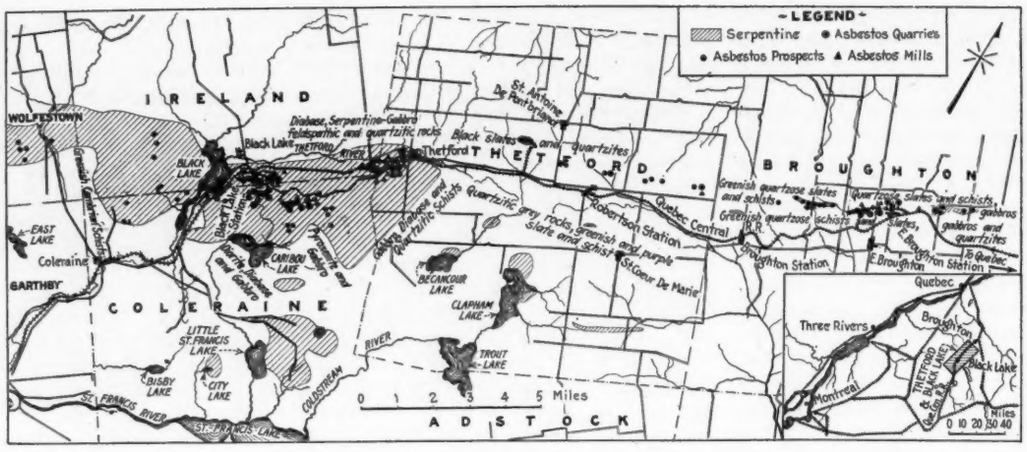


Fig. 5—Geologic map of Quebec asbestos region

to shrinkage stoping at the King mine. He states that underground mining was commenced in February, 1930. To date (November, 1931) eight shrinkage stopes have been completed on the 300-ft. level, 200 to 250 ft. long, 30 to 40 ft. wide, 120 to 150 ft. high. Steel chutes are at 30-ft. centers. A steel-lined manway raise is at each end of each stope. Stopes are at 80-ft. centers. Haulage drifts are 8x9 ft. Electric haulage is used. Chutes are of structural steel and plate, no wood being used underground. Hoisting is by overhead cableway, the cars being run to the track on pit bottom.

The ore is loaded in 5-ton boxes at the chutes, hauled to pit floor, and hoisted. Underground rock the year round costs but little more to mine than open-pit rock. It pays better because barren rock or low-grade ore can be left in place. Accident record in underground work shows a considerable improvement over open-pit work. Steel mine supports are proving satisfactory. On account of higher grades of rock mined, the cost of producing fiber shows a gratifying decrease.

One sublevel stope is being tried, for comparison with shrinkage stoping. Development on the 500-ft. level has progressed to the completion of a main haulage drift 600 ft. long, 8x9 ft. in section, and 500 ft. of crosscuts to stope locations. The present incline haulage and cableway hoisting will soon be replaced by a vertical shaft, Captain Ross states.

The King pit is now about 1,200 ft. long, 1,000 ft. wide, and 400 ft. at its lowest depth. Output has been at a maximum of 7,000 tons of rock per day. Present output is about 2,000 tons. Vertical sinking of down holes for blasting has, wherever possible, superseded face drilling with lifters. These holes, vertical or slightly inclined toward the face, average nearly 40 ft. The 75-ft. holes at the Quebec Asbestos property at East Broughton, described in my 1922 articles, are still being used. The Johns-Manville and the Bell mines, as well as the King, have adopted deep-hole drilling, the holes varying from 32 to 40 ft. deep. At other mines, short holes are still employed, drilled by light hammer drills to a depth of from 10 to 16 ft., and placed along the face of the bench to be blasted. Costs vary widely. My own figures are shown in the table annexed. Captain Ross, in "Chrysotile Asbestos in Canada," recently published by the Department of Mines, gives the cost of drilling alone at from 3 to 17c. per ton of rock broken.

In the present trend toward deep-hole drilling, heavy hammer drills are utilized, mounted on portable derricks. The starter is $3\frac{3}{8}$ in. diameter, for $1\frac{1}{8}$-in. steel to follow. Successive changes are $\frac{1}{2}$ in. each. Holes are spaced close together, not over 5 ft. apart and from 10 to 25 ft. back from the face. Toe holes of equal depth are drilled with the same machine, using drifter mounting. Explosive amounts to 30 to 50 lb. of 40 per cent dynamite. The bottom third of each hole is filled with 2x16-in. sticks of 75 per cent gelatine dynamite. Two caps are affixed to one stick in each hole, the wires leading to these being connected in a parallel series averaging 40 caps to each. Blockholing is done by plugger drills or mud capping. The average breakage in my experience was about 10 tons per pound of explosive; average for the entire district will be nearer 8 tons. Captain Ross gives the cost for explosives at 4 to 7c. per ton of rock quarried.

The two new cableways installed in 1925 at the King pit each have spans of 1,400 ft. They lift a net rock weight of 10 tons, and can operate to a maximum depth of 700 ft. The cubic capacity of the ore box is 220 cu.ft., and it weighs 4,400 lb. Total suspended load at the center is 36,000 lb.; trolleying speed, 1,500 ft. per minute; hoisting speed, 500 ft. per minute, hoist motor, 450 hp.; main cable, 3 in. diameter, 600 r.p.m., 2,200 volts, 3 phase, 60 cycles, full magnetic control; hoisting ropes, $\frac{7}{8}$ in.; stress in main cable, 240,000 lb. Head and foot towers are each movable, as are the receiving bins, of steel, with separate compartments for ore and waste. From these compartments the cars are loaded through hopper bottoms by gates controlled by compressed air, and hauled to mill or waste by electric locomotive. Men employed in mine and mill total 525 under normal conditions.

The Bell mine lies to the southwest of King and directly adjacent to it. In recent years a working agreement between the two companies resulted in the mining of the wall of rich rock dividing the two properties. The two pits therefore now appear as one, except for the difference in maximum depth. That of the Bell is 220 to 240 ft., with a length of 1,200 ft. and a width of 600 ft. It in turn lies to the north and west of the Johnson pit. When the wall separating these two, now being mined by mutual agreement, is removed, one huge opening will result, but controlled by the three companies, with a length of over 2,500 ft. and a width of 1,200 to 1,400 ft.

The Bell now covers an area at the surface of over 15 acres. As explained previously, 40-ft. vertical holes are drilled for blasting. Cranes are used to handle the boxes of waste and mill rock into the cars. One steam shovel loads cars direct. The Bell still has a high value in crudes, which perhaps dictates the use of these methods. Steam locomotives haul the cars to the mill, over standard-gage tracks to the foot of an inclined haulage tunnel, which goes under the mainline of the Quebec Central Railway and the town of Thetford. It is 1,100 ft. long, 12x13 ft. dimensions, and is concreted for nearly its entire length, with a slope varying from 9.86 to 11.22 per cent. The hoist is a Canadian Ingersoll-Rand, equipped with 250-hp. motor. A second drum can readily be adopted for vertical hoisting. At the crest of the incline, 400 ft. from the portal, a steam locomotive picks up the train for haulage to the bins. Average number of men employed under normal conditions is 400, for a mill capacity of 1,500 tons per day of live rock.

The Johnson's Thetford pit is rectangular, 800 ft. long, 500 ft. wide, and now 225 ft. deep. Cableways have been

abandoned, and the system of mining is now similar to that of the Bell—crane loading and incline hoisting, but direct to the mill. Loading is by hand into 2½-ton steel boxes. Here, also, the method is still dictated by the presence of crudes. The boxes are handled by two 15-ton Orton electric cranes into 10-ton steel cars, hauled by two 12-ton Vulcan gas-electric locomotives to the foot of the haulage slope. Constructed by the Foundation Company, of New York, the tunnel is 1,045 ft. long, single track, with a central bypass, 150 ft. long, each side tunnel being 11 ft. wide, the rock-pillar in the middle having a maximum width of 11 ft. The hoist is also 96x32 in., but double-drum equipped with a 450-hp. motor. It will hoist two 10-ton cars of the Granby type on each rope, in balance, at a speed of 750 ft. per minute. Pumping equipment, automatically controlled, consists of a 1,000-g.p.m. centrifugal directly connected to a 200-hp. motor. Present capacity of mill is 2,000 tons per day. Total men employed number 300.

The Beaver pit is to the southwest of the Bell and directly adjacent to the Consolidated, which is separated from it on the north by a thin party wall. Both properties now belong to Asbestos Corporation. Both are being handled through a new inclined-haulage slope. The Beaver constitutes the most southerly portion of the Thetford fiber zone as at present explored. Loading by hand methods, accompanied by cableway hoisting, has been abandoned. A Bucyrus-Erie, 1½-ton electric shovel loads directly into 10-ton steel cars. Electric-trolley locomotives haul the cars from the pits to the foot of a 967-ft. two-compartment inclined tunnel driven on a 20-deg. slope. Each compartment had a width of 11 ft. and a height of 9 ft. in the clear incline of the Johnson. Bad ground was encountered during the driving of this tunnel. A serious fall of rock occurred, and local labor is said to have refused to continue the work. Arrangements were made with the Foundation company to finish it. At the foot of the incline the switches spread to a three-track section, each of which can hold six cars fed by gravity. In the open pits are battery locomotives, or a cable-reel extension to the trolley locomotives. Cars are of Easton type, 42-in. gage. The electric shovel is supplied with current by a rubber-covered cable from an underground substation. The hoist, a Flory, can be operated in or out of balance. Weight of two cars and their 10-ton loads is 32 tons. Total number of men averages 275 for a live-rock mill capacity of 2,000 tons per day.

The Maple Leaf, Vimy Ridge, Asbestos Mines, properties of the Corporation and the Quebec Asbestos Corporation (Philip Carey)—the two latter working on slip fiber—are all operated by small steam shovels mounted on caterpillars. Each loads directly into a car that is hauled up an incline to the head end of the mills. At Asbestos Mines, Ltd., the primary crushers and dryers are at the mines. The rock is then transported by rope tramway 4,800 ft. to the mill proper.

In 1929, a daily average of 6,307 tons was mined in the Johns-Manville pit by the spiral track system, using power shovels and locomotive haulage direct to mills. Of this tonnage a daily average of 4,550 was milled. Present equipment at this mine, known as the Jeffrey, is sufficient to handle 8,000 tons. The pit, almost circular, has a diameter of over 2,000 ft.; maximum depth, 200 ft.; area nearly 75 acres; and track mileage in the pit, over 6½. Steam shovels are being replaced by electric shovels. In 1929 the first step of the electrification consisted in the installation of two Marion No. 4160 shovels on caterpillars, each with 4-yd. bucket. Three 60-ton electric locomotives were also put into operation, in addition to the 15 steam locomotives; 15 air-dump cars of 30-ton capacity were added; 2,000 ft. of diamond drilling blocked out additional ore reserves.

Overburden varies from 20 to over 125 ft. For stripping, 2½-yd. steam shovels of the railroad type are used, equipped with extra long booms and dipper sticks. These load into air-dump cars of 16- and 20-cu.yd. capacity. Trains of ten to fourteen cars are hauled by steam locomotive to a waste dump two miles distant. Dumps, 10 to 90 ft. high, are trimmed by mechanical spreader; tracks are shifted mechanically. Benches are carried at 25 ft. heights—the most efficient height with the rock to be handled.

The semi-portable derrick-mounted system of drilling rig was developed at the Jeffrey mine, where hollow steel is used, 1½ in. diameter by 40 ft. long. A special steel to withstand 1,200 blows per minute had to be developed. Three working benches provide for average production.

In all are four steam shovels, each of 2½-yd. capacity, and two 4-yd. electrics; 100 air-operated dump cars, 16 and 20 yd. capacity, are hauled by eighteen steam locomotives, in addition to the three new electrics. Material handled per day, including overburden, waste, and mill rock, totals 15,000 tons. Of this only 6,000 tons goes to the mills. Waste is nearly barren serpentine, granite, and aplite. Classification of mill and waste is done by shovel operators. Nearly 1,000 men are employed on all the company's Canadian operations—which include mining, milling, and manufacturing.

In 1919, an average of 3,567 men were employed by all the companies in mines and mills in Quebec; 3,082,384 short tons was mined, of which 2,636,788 tons went to the mills. Therefore less than 13 per cent was waste. In 1929 over 29 per cent was waste.

For each worker in mine and mill in 1919 864 tons of rock was mined; or at 275 days—extreme winter conditions and religious holidays cut the working year in Quebec—the tonnage per day per man was 3.14. In 1929, for a daily average of 3,194 men, 6,208,970 tons was mined. So, for each worker, 1,935 tons of rock was mined during the year. Using the same factor of 275 working days, the result indicates that 7.04 tons of rock was handled per day per man. The improvement in efficiency is thus about 225 per cent. In 1928, taken as the last normal year, 5,171,060 tons of rock was mined in Canada by all companies, of which 4,118,044 tons was milled; 3,170 workers, including office staff and engineers, were employed. Thus 1,630 tons was mined per year per man, and 1,300 tons milled. For 275 working days, this gives: 5.93 tons of rock per man per day mined; and 4.73 tons of rock per man per day

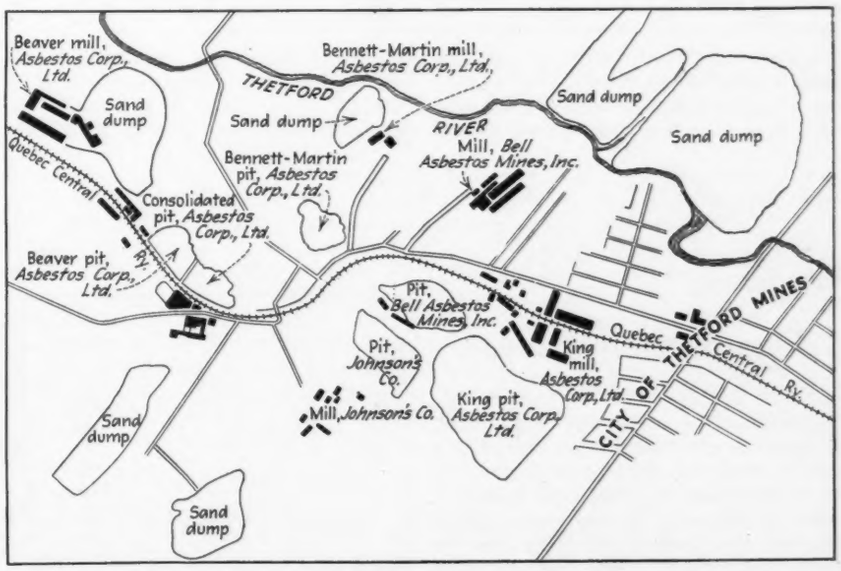


Fig. 6—Sketch map, showing position of asbestos properties at Thetford

Elements in the Cost of Mining

Estimated, Approximate Only	Steam Shovel and Locomotives Per Ton of Rock	Steam Shovel and Locomotives Per Ton of Asbestos	Cableways and Cranes Per Ton of Rock	Cableways and Cranes Per Ton of Asbestos	Shrinkage Stoping Per Ton of Rock	Shrinkage Stoping Per Ton of Asbestos
Prospecting and development	$0.03	$0.05	$0.03	$0.50	$0.50	$8.33
Stripping	0.02	0.25	0.02	0.25	.....	.....
Drilling	0.03	0.50	0.08	1.50	0.17	2.85
Explosives	0.03	0.50	0.05	0.80	0.08	1.33
Sorting and loading	0.10	1.70	0.60	9.60	0.20	3.34
Transportation	0.10	1.70	0.10	1.70	0.10	1.67
Hoisting	.....	.....	0.05	0.70	0.03	0.50
Waste-dumps	0.005	0.08	0.005	0.08	0.005	0.08
Pumping	0.003	0.05	0.003	0.05	0.01	0.16
Safety, lighting, incidentals	0.002	0.03	0.002	0.03	0.002	0.03
Total mining cost	$0.320	$5.31	$0.940	$15.31	$1.097	$18.29
Power and fuel	0.227	3.63	0.227	3.63	0.227	3.63
	$0.547	$8.94	$1.167	$18.941	$1.324	$21.92
Milling, bagging, loading	0.50	8.00	0.50	8.00	0.50	8.00
Total cost	$1.05	$16.94	$1.67	$26.94	$1.82	$29.92

milled; 273,865 tons of asbestos made—86.4 tons per man per year, or 0.314 tons per man per day. Each man produced asbestos worth $3,550, against which he was paid $1,260. His average daily earning was $4.57 for ten hours, or $0.457 per hour. The rock as mined was worth $2.18 per ton in the ground. This rock averaged 2.54 per cent asbestos through paper stock, 5.28 per cent through shorts. Electric energy cost $0.011 per kilowatt-hour, and 12 kw.-hr. was used for each ton of rock mined and milled. Cost of electric energy and fuel was $0.227 per ton of rock mined; cost of labor, $0.773 per ton of rock mined; cost per ton of rock mined and milled, $1 for labor, fuel, and power. This is about $18.95 per ton of asbestos (all grades) produced. Labor, fuel, and electricity absorbs about 60 per cent of all costs, so the total cost per ton of rock mined and milled would be about $1.67, and the cost per ton of asbestos about $31.70 per ton (all grades), exclusive of service or capital investment, interest, sales cost, and sundry other miscellaneous overhead expenses.

The cableway cost of mining per ton of rock, $1.67, agrees with the figures taken from actual operations. Discrepancy between the actual estimated cost per average tons of asbestos in 1928 in Quebec and the foregoing figures is caused by the fact that in the accompanying calculations I have assumed a 6 per cent content in the rock, whereas the actual percentage for all grades in the 1928 figures was 5.28 per cent. Repairs, maintenance, insurance, amortization, interest charges, general management and head office expense, sales expense, and other charges have not been included. The steam-shovel mining and milling costs are almost identical with those experienced by me in the Broughton district of Quebec; the cableway and crane costs to those at Black Lake; shrinkage-stoping costs to those at Thetford. I believe the estimates thus check actual operating figures closely.

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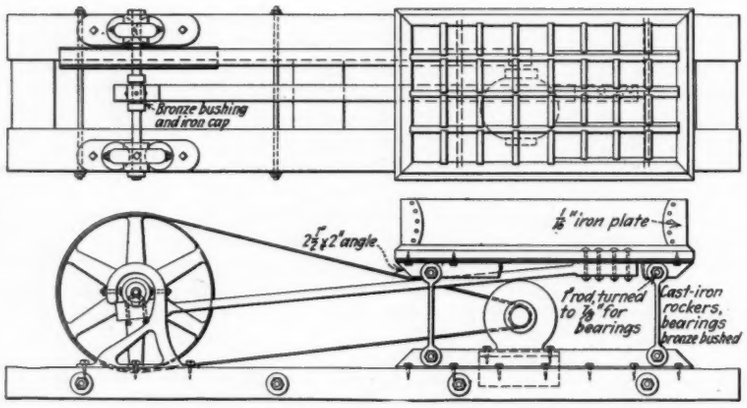
Laboratory Shaking Machine


Bronze bushing and iron cap
2½"x2" angle
⅛" iron plate
Rod turned to ½" for bearings
Cast-iron rockers, bearings bronze bushed

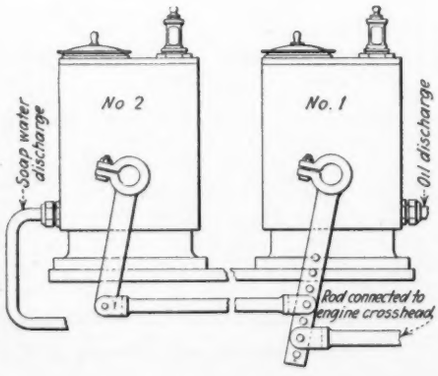
A 32-compartment shaking machine illustrated in the accompanying sketch, has given satisfactory service for leaching tests in the laboratory, writes Nathaniel Herz, chief chemist of Homestake Mining Company, Lead, S. D. It has sixteen compartments 4x4 in. and sixteen 3.5x3.5 in., all 6 in. deep, with lineoleum-covered bottom. For further protection the bottles are wrapped in cloth. The machine is operated by a ½-hp., 1,750-r.p.m. motor, belted to a flywheel and 1¼-in. crankshaft, the center of which imparts an oscillating motion to the table by a 1x2-in. connecting rod of oak, at 2½-in. stroke.

The machine is used for all amalgamation tests, and for nearly all cyanide work. Agitation has been found more satisfactory for coarse sand than small-scale percolation tests; sands are difficult to charge without channels, and good aeration is also almost impossible to obtain. Agitation for two or three days usually gives maximum obtainable extraction for almost any sand. In the case of slimes, the time of agitation generally varies between two hours and eight hours.

■ ■ ■

Cleaning Compressor Valves

FREQUENT cleaning with soap water is essential to keep compressor valves free from carbon and dirt. This is usually accomplished by feeding a solution of soap for a short period through the mechanical oiler on the compressor, taking good care to clean the oiler well before and after replacing the oil with soap. To avoid waste of time required for this procedure, A. Y. Smith, master mechanic at the Pilares mine of Moctezuma Copper, Sonora, Mexico, attached to the engine frame two mechanical oilers in tandem, as shown in the accompanying sketch. Oiler No. 2, containing the soap water, is actuated by a small connecting rod bolted onto the lever of oiler No. 1, operated from the engine crosshead. Feed of soap water or oil to the valves is controlled by two small valves on the oiler-discharge pipes.



Mine-Shaft Equipment—I

Lucien Eaton

SHAFTS are used in opening, developing, and operating underground mines. Their location, size, type, and construction vary with the uses to which they are put. Even at open-pit mines, if the orebody extends below the natural drainage level, shafts may be necessary for pumping, and for hoisting as well.

Geological conditions may be such that exploration by diamond drill or churn drill is impossible or unsatisfactory, and exploration shafts must be used. These, usually small and inexpensively equipped, are not expected to be maintained for long after mining starts. On account of their short life they are not to the same degree subject to the limitations as to location, type, and construction as other shafts that may be required. They are usually sunk in or near the ore, and may be irregular in inclination, and even in direction, the exploration of the ore having first consideration.

Shafts are used for ventilation and pumping, for hoisting waste and ore, and for hoisting and lowering men and supplies, as well as for exploring. All of these functions may be performed in one shaft, but in most states a second outlet is required by law at a mine employing more than twenty men. If hoisting operations and the handling of men and supplies are concentrated at one main shaft, this second shaft would be used only for ventilation and as an exit in case of fire or accident underground. It should be far enough away from the main shaft to avoid being cut off by the fire or accident that might put the main shaft out of commission. The same considerations as to safety and permanency apply to it as to the main shaft, but to a somewhat lesser degree, because the investment in shaft and equipment is less. Furthermore, it is not subject to the same limitations imposed by the necessity for convenient connection with transportation facilities or with the rest of the mine plant.

The advantages and economies in investment and operation to be derived from concentrating all hoisting operations at one well-equipped, centrally located shaft are becoming generally appreciated, and the custom of sinking several shafts at every mine, prevalent a generation or two ago and still existing in some districts, is not now as common in the United States. Appreciation of the fact that a thorough preliminary study and investigation should be made before sinking starts, and that money spent in such an investigation is amply repaid, is also gaining ground.

Selection of the right size and type of shaft to serve as the main hoisting shaft, and the choice of the best location with respect to the orebodies to be mined and to the surface works of the property, are important. The size and type of shaft best suited to the exploitation of the property may be more easily determined than its best location, because the capacity required and the methods to be followed in development, mining, and transportation can be determined; but the factors affecting the selection of the location are not often so definitely known. Available sites may be so limited by topography, property ownership, transportation facilities, or geological conditions that little choice is left to the management, and the question becomes more a matter of making the best of conditions than of choosing an ideal situation. Two or more alternatives are nearly always available, and the advantages and disadvantages of each must be carefully weighed before a decision is reached. This paper will review the factors affecting the selection of the right kind of shaft and its proper location, and indicate the methods that may be followed in reaching a decision. To appreciate the importance of this matter one need only review the great number of shafts that have been wrecked by subsequent mining operations, through errors in location; or have been replaced because of inadequate capacity or faulty design. Such examples can be found in nearly every camp where large orebodies have been mined.

The discussion that follows regarding the relative merits of different locations for shafts, and of different shapes and arrangements and types of construction, and of inclined and vertical shafts, is not to be construed as applying to exploratory or other temporary shafts, which are expected to be lost in the process of mining. Nevertheless, the arguments presented apply to shafts of this character and purpose, in so far as the usefulness of these shafts would not be impaired by following the principles laid down.

To start with an ideal situation, having no complications, is desirable in discussing the subject of the proper location of shafts; later to consider what effect such complications might have upon the choice of site. Assume, therefore, that the topography presents no difficulties as regards drainage, transportation, construction, or plant layout, and that no geological complications are in evidence, such as quicksand, weak or water-bearing strata or similar phenomena. Assume also that the dip, pitch, and approximate size of the orebody are known. Under these conditions, to equalize the distances between the shaft and parts of the orebody, the shaft should be sunk as nearly on line with the center of gravity of the orebody as is convenient and safe. Safety, of paramount importance, should seldom yield precedence to any other consideration.

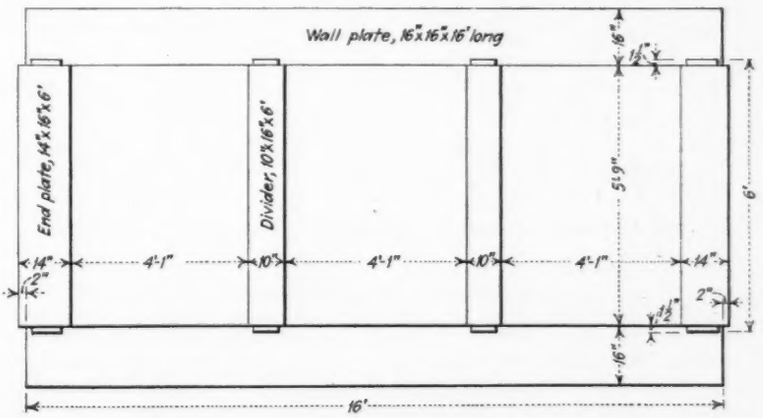


Fig. 1—Argonaut shaft (after I. C. 6311, U. S. Bureau of Mines)

Geologically speaking, three locations are available for the shaft—(1) In the hanging wall; (2) in the orebody or vein; (3) in the footwall.

A shaft sunk in the hanging wall through the vein has the advantage of decreasing the amount of crosscutting necessary to reach the ore, but this advantage is obtained at the expense of safety. Except in rare instances, to choose a location of this sort, if it can possibly be avoided, is seldom wise. Mining a thin and reasonably flat bed may be considered an exception, and in such operations the selection of a

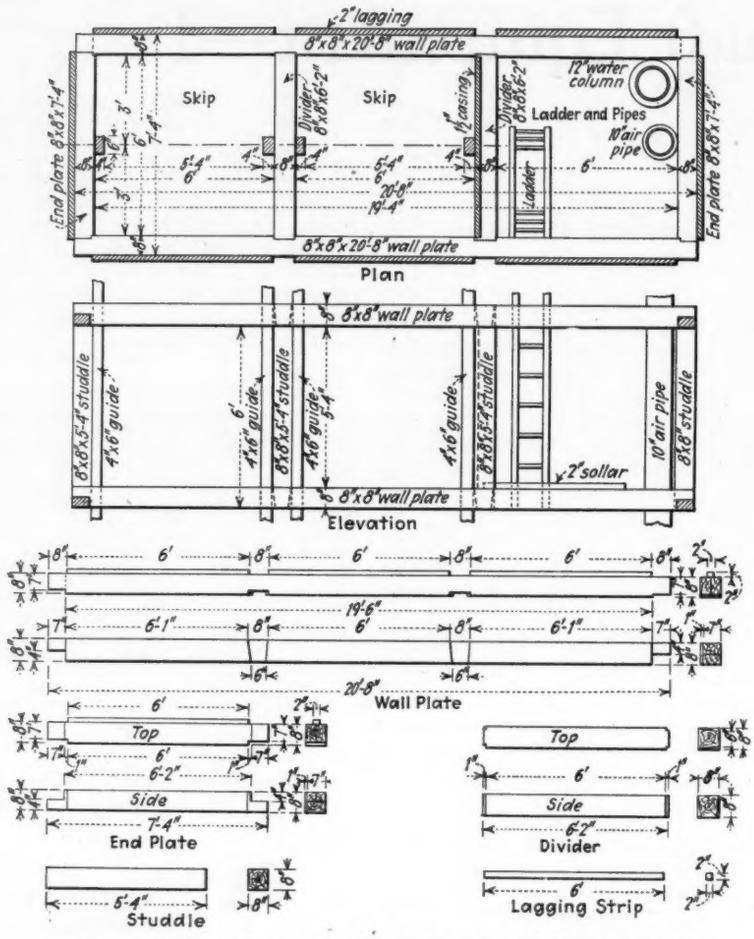


Fig. 2—Details of shaft No. 1, Chambisi mine, Southern Rhodesia

hanging-wall site may be governed by limitations of ownership, as in the deep-level mines of the Rand.

Moreover, flat-lying deposits, apparently, like those in the southeastern Missouri lead district, which have strong ore and a strong hanging wall, and in which, during the process of mining, sufficient pillars are left in place to prevent ground movement, might also be considered to be exceptions; but even here a wiser course is to have the shaft off at one side far enough to be safe from the effects of any possible cave-in. In any event, in order to be assured of even temporary security, a considerable tonnage of ore must be tied up in pillars, and a hazard is involved that this ore will eventually be lost. Except in rare instances the removal of the ore causes subsidence; and filling, even sand laid down with water, seldom prevents it completely, although limiting the amount.

In the days before mechanical haulage, when the endurance of a man's legs set an arbitrary limit to the distance that ore could be economically trammed, and when progress in drifting was a small fraction of present accomplishment, the consideration to be given to the reduction in the length of crosscuts afforded by hanging-wall shafts was much more important than should be the condition now. It probably accounts for the large number of hanging-wall shafts that were sunk a generation or two ago.

The same arguments that have been used against hanging-wall shafts apply to shafts sunk in the orebody, and the amount of ore tied up in shaft pillars is likely to be as large or even larger. In a small orebody the exploration value of a shaft sunk in the ore may be sufficient to warrant the selection of such a location, but this is usually open to question, and has no proper bearing on this discussion.

A shaft sunk in the ore may be justified for financial reasons, if it be to reduce initial capital expenditure or to put the mine on a paying basis quickly. This may be desirable to avoid borrowing money or to take advantage of a favorable market. But these are special conditions, which have only a temporary bearing on the situation, and do not affect the principles involved. In such circumstances the shaft would have to be replaced later. In a large majority of instances the best location for a shaft is in the footwall. Here it is safe, and safety is paramount. But it must be far enough back in the footwall to be beyond the reach of subsidence cracks caused by removal of the orebody.

In many instances, if not in the majority, the angle of break, if the bedding planes do not constitute lines of weakness, is steeper on the footwall side of an orebody, where the breaks are more nearly parallel to the bedding, than on the hanging-wall side, where they are across it. To allow an angle of 70 deg. from the nearest or lowest point of the orebody, whichever is the more effective, as the angle of break in rock, is usually considered safe; and an angle of 45 deg. for sand, gravel, or loose formations. These angles are fairly safe in homogeneous formations not weakened by faults or pronounced systems of joints, if the depth considered is not too great. A definite limit cannot be set for the depth, to which the angle will apply, because so much depends on the strength of the rock. In strong formations, 70 deg. may be a safe angle to use for

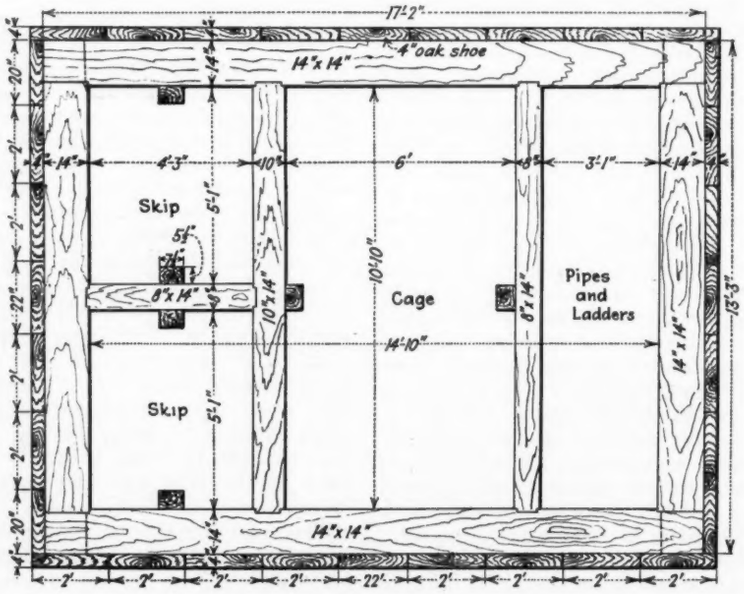


Fig. 3—Plan of timbering, standard shaft, Cleveland-Cliffs Iron Company

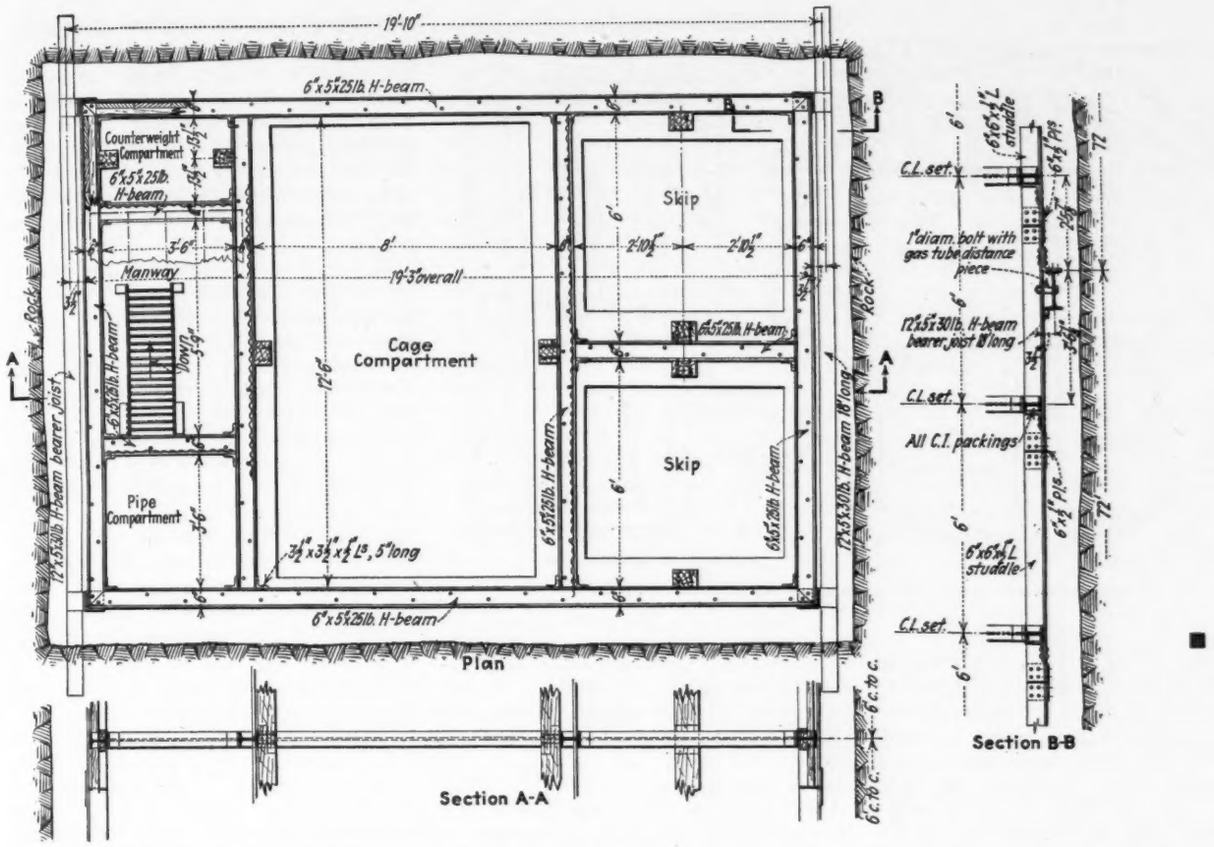


Fig. 4—Steel shaft set, Roan Antelope mine, Southern Rhodesia

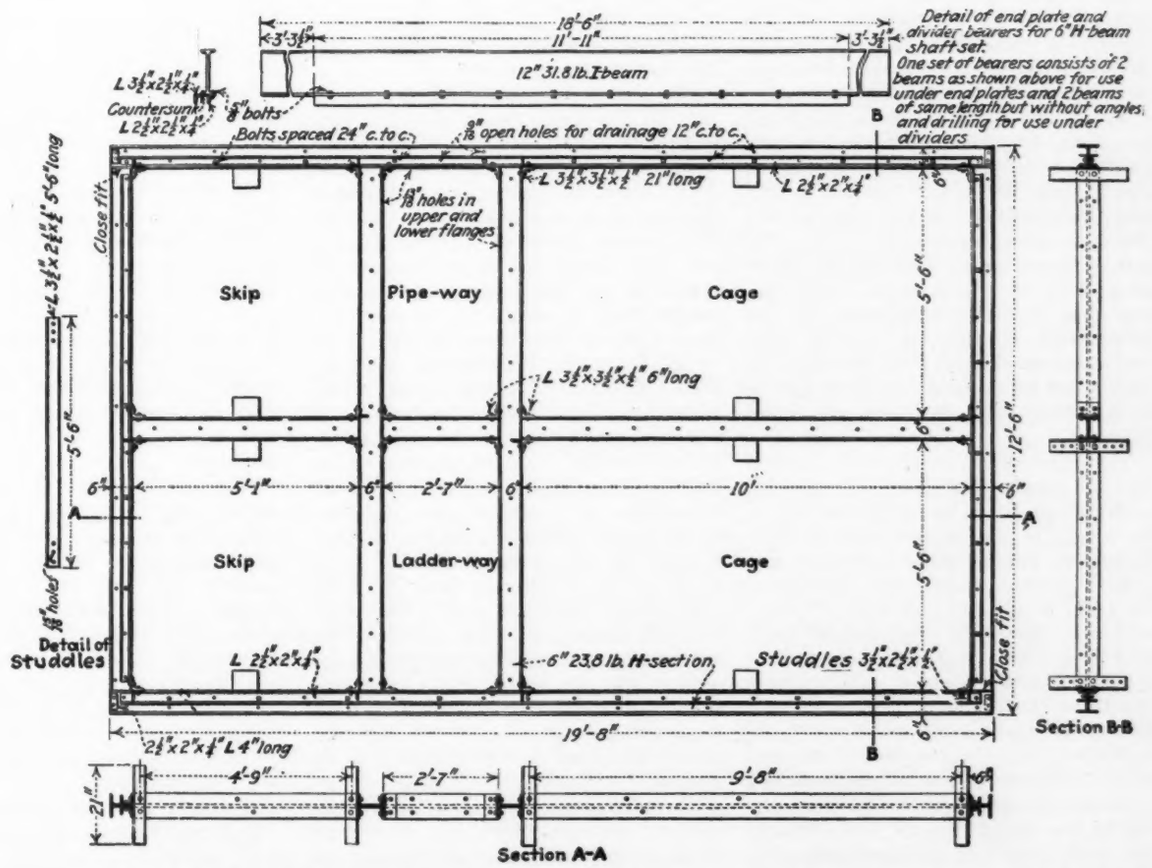

18'-6"
11'-11"
12" 31.8 lb. I beam
Detail of end plate and divider bearers for 6" H-beam shaft set. One set of bearers consists of 2 beams as shown above for use under end plates and 2 beams of same length but without angles and drilling for use under dividers
Countersunk
⅞" bolts
Bolts spaced 24" c. to c.
⅞" open holes for drainage 12" c. to c.
Close fit.
Skip
Pipe-way
Cage
Skip
Ladder-way
Cage
Detail of Studdles
6" 23.8 lb. H-section
Studdles
19'-8"
Section A-A
Section B-B

Fig. 5—Steel shaft set, Eureka mine, Gogebic Range, Mich. (A. L. Foss, *Eng. & Min. Jour.*, Sept. 29, 1928).

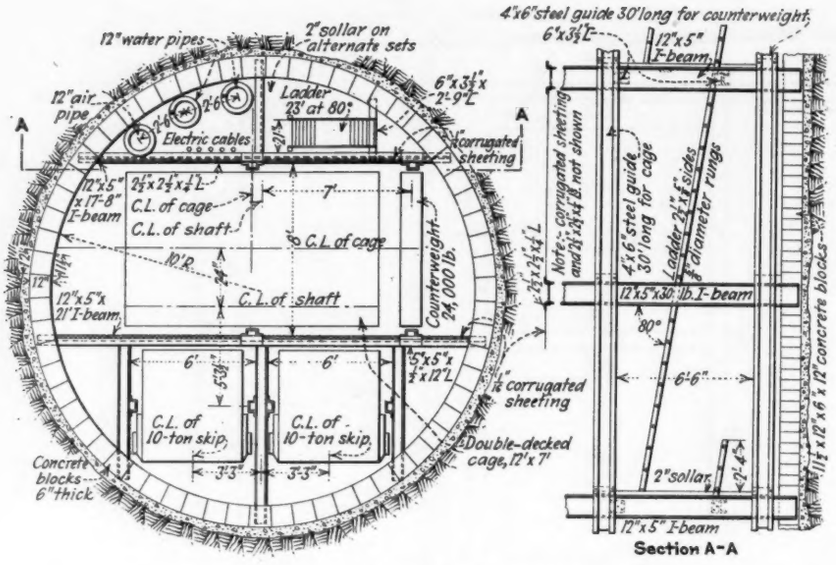


Fig. 6—Circular shaft, Mufulira mine, Northern Rhodesia

depths up to 500 or even 1,000 ft. An angle of 60 deg. is more conservative. The greater the depth the flatter should be the angle, because rock in large quantities acts much the same as sand or gravel. The influence of fault planes, slips, and other lines of weakness should not be overlooked. On the hanging-wall side a flatter angle of break may be expected, and under extreme conditions I have known it to be less than 30 deg. On the average, an angle 20 deg. less than on the footwall side may be considered safe; but this depends a good deal on the formation overlying the ore, as some formations have a very steep angle of break.

When the probable angle of break has been determined, the area of subsidence over the orebody should be plotted, and the shaft should, if possible, be located outside this area, usually 100 ft. or more beyond the limits. If it can be placed nearly opposite the center of gravity of the orebody, the transportation of ore underground and the distribution of supplies can be most easily balanced, but this consideration should not be given as much weight as some others that will be mentioned later.

The chief disadvantage of a vertical footwall shaft, especially one serving orebodies of fairly low angles of dip, is the length of the crosscuts required at depth. In mining small orebodies, this is an important consideration, both from the point of view of cost and from that of time required to open a new level. The time element is not as important as it was. The technique of rapid drifting has been brought to a high degree of perfection; and, in the mining of large orebodies, the cost per ton is not materially affected by the longer crosscuts.

Of the complications that have a bearing on the choice of shaft site, probably the most important is topography. To find shaft sites that are entirely satisfactory in hilly or mountainous districts may be impossible. A suitable site must be found for the engine house, far enough away from the shaft to allow a proper fleet angle for hoisting ropes. Room must be provided for the other mine buildings, and these should not be too close to the shaft, on account of fire hazard, and for the storage and handling of supplies. At mines using much timber underground, a convenient timber yard not too far from the shaft is important. Facilities must be available for the disposal of waste. And, above all, proper facilities are necessary for the transportation of ore.

At mines where the crude ore is concentrated in a mill on the property some leeway is allowable; but at properties where the product must be loaded directly into railroad cars the proper location of the railroad track may be the determining factor in deciding location for the shaft. The advantage of moderate and favorable grades, and of curves of large radius in the standard-gage railroad tracks on the surface, is so important that it outweighs any ordinary underground complication in arrangement, and often indicates the advisability of bringing the shaft to the railroad rather than the railroad to the shaft. This applies especially to the orientation of a vertical shaft, the desirability of having the loading pockets parallel to the railroad tracks being apparent. In an inclined shaft, the problem is more difficult to solve. As underground tracks are not affected by topography, and as their narrower gage permits the use of curves of relatively small radius, to accommodate their direction and arrangement to the orientation of the shaft is easier than it is on surface.

The character of the geological formations through which the shaft must be sunk has an important bearing on the choice of shaft site. Quicksand, running ground, or water-bearing strata should, if possible, be avoided, even at the expense of much longer crosscuts, for the delay and expense of sinking in such formations will pay for a tremendous amount of drifting. Preferably the shaft should be sunk in strong and solid rock without excessive overburden and with as small a flow of water as possible.

Not infrequently the ore formation is porous and is saturated with water, and the footwall is comparatively dry. Under such conditions, to sink the shaft in the dry formation is good policy, and not to open the wet ground until pumps have been installed.

In balancing the importance of the various considerations that bear upon the question of a proper shaft site, a certain amount of compromise is necessary, and no hard-and-fast rule, except one, can be laid down. That rule is that the shaft must be safe. The other considerations must be left to the judgment of the engineers planning the work, and the relative importance of such considerations will vary in every instance.

The next question to be settled is the type of shaft to be used, whether inclined or vertical. From an operating standpoint the vertical shaft is to be preferred. For equal sizes, its capacity is greater, not only because higher hoisting speeds are possible with safety, but because the clearances in the hoisting compartments are smaller, and skips of greater capacity can therefore be used. Furthermore, maintenance charges are materially less. Records made several years ago in the Lake Superior iron country showed maintenance charges of inclined shafts to be two and one-half times as great as those of vertical shafts.

When the orebody dips at an angle of 70 deg. or less, a saving in first cost may be effected by sinking an inclined instead of a vertical shaft. If the inclination of the shaft is made parallel to the average dip, the length of the crosscuts will be minimized, but this saving in crosscutting is partly offset by the increase in the length of the shaft, and the net saving is not as great as it would seem to be on first consideration.

The relative costs of shaft sinking and drifting vary with local conditions in any district, and vary as between districts, so that no definite comparison is possible that will be universally applicable. Costs of shafts run from six to ten times the cost of drifts; and, if an average of eight is assumed as a basis for calculations, the percentage of saving in first cost to be effected by the use of an inclined shaft, except at very flat dips, is comparatively small—in fact, insufficient to offset the higher maintenance and operating costs.

Compound shafts—shafts which change from vertical to inclined—have been used, but they cannot be recommended.

Operating and maintenance costs are high, capacity is relatively small, and they introduce complications which interfere seriously with effective work.

The shape of the shaft and the size and arrangement of the compartments is the next subject to be considered. Most inclines are rectangular in cross-section, and usually consist of several compartments arranged side by side on the line of the strike. A few examples may be cited, however, where, to permit the use of a larger cage, the skip roads were placed on the hanging-wall side, and the pipe and ladder compartments were under them, but this arrangement was not entirely satisfactory. An illustration of a successful incline is shown in Fig. 1.

Vertical shafts may be rectangular or circular in cross-section. Oval sections have been used occasionally, but are rare. Rectangular shafts are most common in the United States, and circular shafts in Europe; but the popularity of circular shafts is increasing in the United States, and their use will probably be more common as improved methods of sinking and lining them are adopted. The chief objection has been their higher cost of construction. Rectangular shafts are usually long and narrow in cross-section. This design, on account of the short dividings, has the advantage of being able effectively to withstand heavy rock pressure. It has the disadvantage, however, of inhibiting the use of a large cage. Shafts of this type are in use on the Witwatersrand, having as many as seven compartments in line. A recent shaft of this type is shown in Fig. 2.

Rectangular shafts of shorter and wider cross-section are often incorrectly referred to as "square shafts." This shape is increasing in popularity, and many of the new shafts have been ot this design. Usually two skip roads are put side by side across one end, and one or two cage compartments in the middle, and the pipe and ladder compartments at the end opposite the skips; or two skips may be arranged at one end, two cages at the other, and the pipe and ladder compartments in the middle. The wide shaft makes possible the use of a cage large enough to carry a locomotive or tramcar, or a timber truck with a load of mine timber on it. It will not resist ground pressure as well as the long and narrow type, but it contains no waste space and has definite operating advantages. Examples of shafts of this cross-section are shown in Figs. 3, 4, and 5.

Circular shafts have the advantage of being able effectively to withstand heavy ground pressure. Being usually lined with incombustible material, they are

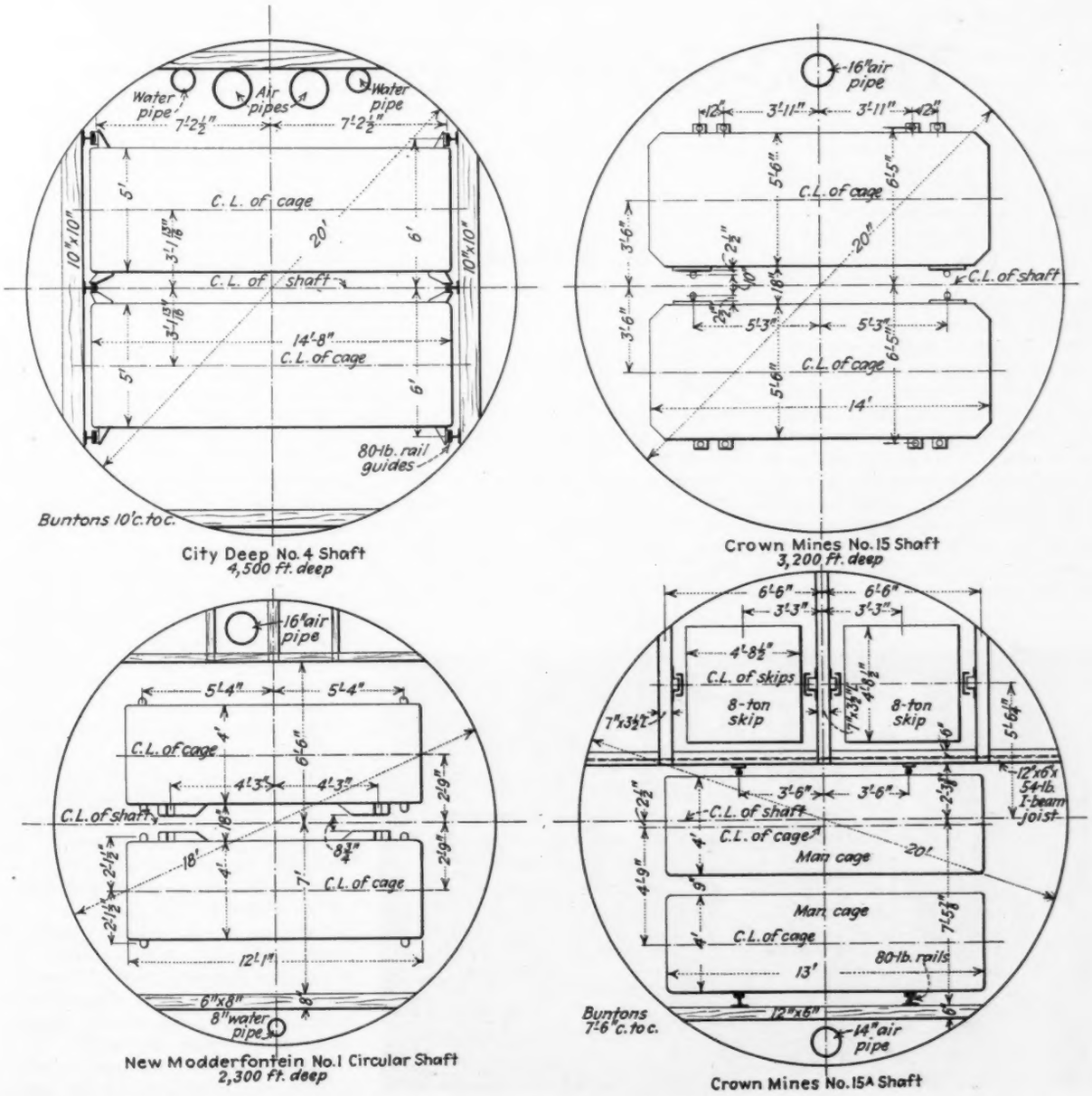


Fig. 7—Circular shafts, Witwatersrand mines

both fireproof and permanent. They require, however, about 50 per cent more excavation than rectangular shafts of the same capacity. The arrangement of the hoisting compartments is likely to follow closely that of the "square shafts," and the pipes and ladders are placed in the odd spaces left outside the hoisting compartments. At best there is considerable waste space, of value only for ventilation.

Various arrangements of compartments in use on the Rand are shown in Fig. 7, taken from Mr. William Elsdon-Dew's paper on "Winding at Great Depths," read before the Empire Mining and Metallurgical Congress in 1930. Note that the New Modderfontein No. 1 Shaft and Crown Mines No. 15 Shaft use rope guides, and that City Deep No. 4 Shaft and Crown Mines No. 15A Shaft use rails as guides for the cages, and the last mentioned also uses a special rolled steel section for skip guides. This question of guides, however, will be taken up in detail later.

If the shaft has to be sunk in heavy ground, it may be necessary, to assure safety and minimize the repairs needed, to limit the dimensions of the compartments to sizes smaller than good operating practice would indicate, and the skips and cages will then have to be designed to fit the compartments. Under normal conditions, however, the ideas set forth in the succeeding discussion may be used as guides in deciding upon the dimensions to be used.

(To be continued)

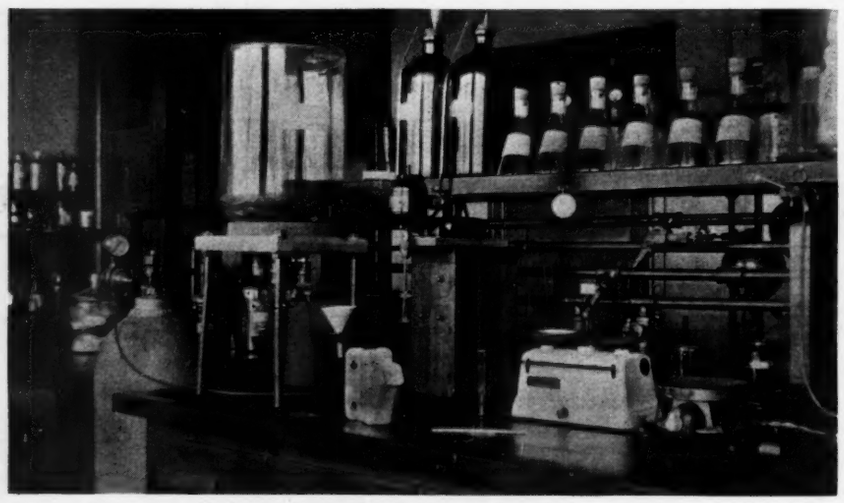
■ ■ ■

Perfection and Use of The 50-Gram Flotation Cell

M. S. Hansen

DEVELOPMENT of the 50-gram flotation cell and laboratory process by J. F. Gates and L. K. Jacobson in 1925 (*E. & M. J.*, Vol. 119, No. 19, pp. 771-772), using pure minerals and synthetic ores, has been of great importance in the study of flotation fundamentals. The cell is large enough to give quantitative results and yet does not require an excessive amount of materials and labor to effect a satisfactory operation. It is especially useful in the study of flotation reagents on pure minerals and is particularly applicable to research work for the development of special reagents.

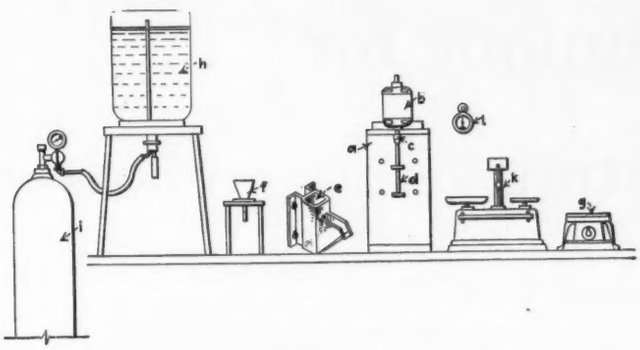
Because of the relatively small amount of mineral and reagent used in a test, the conditions under which the test is run must be closely controlled; for slight variations in the conditions make relatively large differences in the results. The principal source of error has come by contamination of the flotation circuit with the material of the cell itself. Neither the iron nor aluminum cell, the iron cell painted with aluminum paint (*E.&M.J.*, Vol. 119, No. 19, pp. 771-772) nor the celluloid cell (*E. & M. J.*, Vol. 126, No. 13, p. 504) has proved entirely satisfactory in this respect. The iron cell is easily oxidized, the aluminum cell is attacked by alkaline solutions, and both cells are corroded by copper salts and ores. Highly objectionable salts are thus thrown into the circuit, exerting an effect on the flotation reagents or upon the floatability of the mineral itself. Furthermore, metal surfaces of any kind in contact with the flotation circuit are objectionable because they strongly adsorb oily reagents. The iron cell coated with aluminum paint is open to these same objections and in addition some contamination arises from the vehicle in which the aluminum paint is mixed.



Equipment ready for testing

The celluloid cell, though it has some advantages over the metal cell, has many of its disadvantages, and other disadvantages of its own. The aluminum rivets which hold the celluloid sheeting together are a source of trouble. Joints around the rivets and between the celluloid sheets become exposed with use and fill with minerals or reagents. Collodion or bakelite varnish has not been wholly successful in obviating this trouble. Further, and perhaps most objectionable of all, constituents of celluloid, such as camphor, become dissolved in the flotation circuit and modify results materially.

Samples of $\frac{1}{8}$-in. celluloid stock from seven different manufacturers were tested in a flotation circuit to determine the degree of inertness toward flotation solutions. All of the samples showed, at first, an increase in weight ranging from 3 to 15 mg. per square inch, indicating an adsorption of the aqueous solutions, probably by hydration. Upon



Arrangement of flotation test apparatus

(*a*) flotation stand; (*b*) variable-speed motor; (*c*) chuck, for holding impeller; (*d*) glass or cast-iron enameled impeller; (*e*) cast-iron enameled 50-gram flotation cell; (*f*) Büchner funnel; (*g*) electric hot plate; (*h*) mineral bottle; (*i*) nitrogen tank; (*k*) torsion balance; (*j*) stop watch.

air-drying these samples they showed a decrease in weight ranging from 1 to 60 mg. per square inch over the original weight. One experiment in which a nearly new 50-gram celluloid cell was weighed before and after a flotation test showed a decrease in weight of 0.138 grams (equivalent to 5.52 lb. per ton of ore mineral).

A 50-gram flotation cell developed and in use at the Reilly Laboratories obviates all the difficulties incident to the old type cells. This new cell, which follows essentially the design described by Gates and Jacobson, is made up as an iron casting coated with a layer of acid-resisting enamel. A glass impeller or a cast-iron impeller with a vitreous enamel coating is used, so that all of the surface in contact with the flotation pulp is of inert vitreous material, which makes it possible for the first time to obtain true quantitative results from flotation tests.

The accompanying sketch shows a convenient set-up for work on a 50-gram flotation machine. The stand (*a*) is 13 in. high and can conveniently be made from 2x6-in. lumber. The motor (*b*) is a small variable-speed motor manufactured by the Bodine Electric Company, Chicago, and has a maximum speed of 3,400 r.p.m. It is held on top of the stand by a thumbscrew which allows some adjustment of position for the motor. The small chuck (*c*) fastened to the shaft of the motor holds the impeller (*d*), which may be either of glass or cast iron covered with vitreous enamel. A small Büchner funnel (*f*) connected to an aspirator is used for catching and filtering the concentrate. The small-size electric hot plate (*g*) is run at low heat and will dry the concentrate retained on the filter paper in a few minutes.

In operating a 50-gram machine the accuracy and relative value of the results depends upon the elimination of every possible variable excepting the one under consideration. A standardized procedure is to be preferred for all tests. The dilution of the pulp in conditioning, the length of time of conditioning, the speed of the motor, the pH of the circuit, the method of adding reagents, the order of addition of the reagents, and the stage at which the reagents are added must be standardized. Usually to have a fixed length of time for running a test is not so important. The most satisfactory results have been obtained by taking all the concentrate that a given quantity of reagent will cause to float during a reasonable time. This will eliminate the possibility of rejecting, say, selectors that may be very effective but slow in action.

The condition of the mineral must be constantly checked against some high-standard selector reagent such as potassium ethyl xanthate, so as to prevent the use of mineral having a floatability below a set standard. Since floatability is chiefly a question of the extent of oxidation of the mineral, great care must be taken to prevent such change. Mineral allowed to stand in contact with air, and particularly the air of a chemical laboratory, will oxidize so rapidly that after even a week's time its floatability is considerably lowered.

One convenient method of cutting down the oxidation of mineral to a minimum is to keep it in an atmosphere of nitrogen. An arrangement for doing this is also shown in the sketch. As soon as the mineral has been prepared it is placed in a bottle (*h*) and the air thoroughly displaced with nitrogen; then, as the mineral is taken from the mineral bottle, it is displaced simultaneously with nitrogen. Prepared mineral that has become so oxidized that it cannot be used for pure mineral work can be recleaned by a weak acid treatment followed by thorough washing, or frequently by washing alone.

Galena (PbS) is the most satisfactory mineral for pure mineral work in testing out the activity of selector reagents for flotation. It is relatively cheap, easy to obtain, and in general it is very pure. Chalcocite (Cu_2S) will also give accurate and dependable results if it is pure. But chalcocite ordinarily is much more costly and cannot be obtained in bulk with as high a purity as galena. Since sphalerite (ZnS) can be obtained relatively cheaply and of high purity, it represents a mineral that can be substituted for chalcocite in pure mineral work by merely using copper sulphate in conjunction with it. The quantity of copper sulphate must be accurately determined, since an excess, as well as a deficiency, is unsatisfactory; but by its use the sphalerite is superficially changed by a light and fresh coat of copper sulphide to a mineral having a high order of floatability. Pyrite (FeS_2), for the pure mineral work mentioned, is not so satisfactory because of the ease with which it oxidizes.

Frequently it is necessary to test the action of flotation reagents in the separation of one mineral from another, such as the flotation of chalcocite ahead of pyrite, or other combinations. This is conveniently done on the 50-gram cell, using pure minerals. One mineral of the mixture will be sized to, say, minus 65 plus 100 mesh, and the other mineral will be sized to minus 100 plus 320. The concentrate from such a mixture of minerals can, therefore, be analyzed quickly by separating the minerals by screening. It is better to have the most easily floated mineral in such a mixture, the large mesh size.

The utility of the 50-gram cell lies chiefly in the field of research, including the development of new and active flotation selector reagents, and in the mill, where a quick and accurate method of comparing the relative activities of several selector reagents is required.

An acknowledgment is made here for the many valuable suggestions and criticisms that have been made by Mr. O. D. Cunningham, head of the Department of Flotation Research, The Reilly Chemical Company, in the development of this work.

An Efficient Technique for Drilling and Blasting

W. R. Meyers

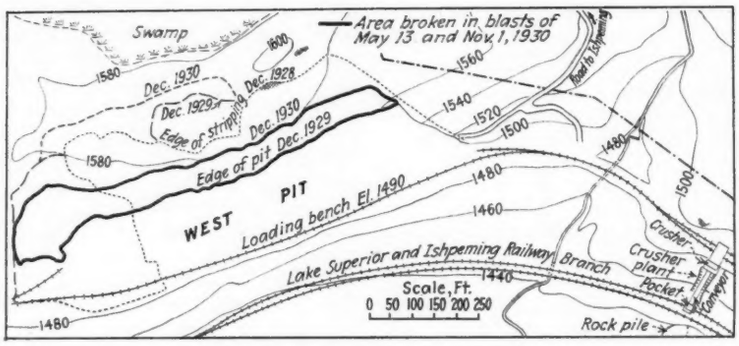


Fig. 1—Part of Tilden surface map, showing area broken in blasts of May 13 and Nov. 1, 1930

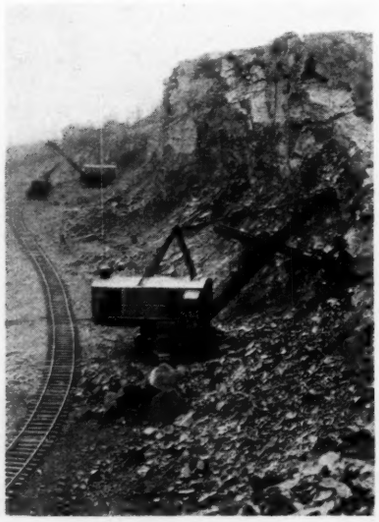
THE Tilden mine, 3 miles south of Ishpeming, Mich., owned in fee and operated by the Cleveland-Cliffs Iron Company, adjoins the Foster mine, formerly worked by the same company, but abandoned 30 years ago. The existence of the Tilden orebody has long been known, but only in the last few years was merchantable ore found. It is a hard siliceous hematite of ideal physical structure, and desired by furnace men because of uniform grade. It averages about 40 per cent iron and 40 per cent silica, and is very low in impurities and moisture.

Preparations to open the pit by the hydraulic stripping of the 2 to 6 ft. of overburden were begun in 1928, when the Lake Superior & Ishpeming Railroad built a spur to the mine site. Mining was started at the opening of the Great Lakes navigation in 1929, with a production of 442,000 tons for the first season. The topography of the orebody lent itself to this method of stripping. The ore, which extends into the hillside, with a swamp to the south, is being mined in two benches, the upper being worked at an elevation of 1,490 ft., or 50 ft. above the main railroad track. The face of the pit is 900 ft. long, and the maximum height of the present working face at the center is 100 ft., about 80 ft. at the east and 60 ft. at the west end.

Average percentage analysis of shipments from the Tilden mine in the season of 1929-30 is as follows: Iron, 40.35; phosphorus, 0.046; silica, 40.25; manganese, 0.08; alumina, 0.96; lime, 0.65; magnesia, 0.40; sulphur, 0.009; loss by ignition, 0.40; moisture, 2.50. The company can also mine a siliceous ore that is very low in phosphorus, averaging approximately 0.015 per cent.

To date the tonnage of ore broken is 880,000. During the first year of mining at the Tilden, eleven blasts were made, compared with two in 1930. The first blast, on April 16, 1929, broke 80,000 tons, which was loaded by the middle of May, when blasting was again necessary. Drilling equipment at that time consisted of three Sanderson Cyclone churn units, which were unable to drill enough holes to supply the shovels, which, during the first year, loaded ore as fast as it could be drilled and blasted, and frequent blasts had to be made to keep broken ore ahead of the shovels. To overcome this difficulty, the operators decided to drill during the winter months. Two rows of holes were drilled along the entire face of the pit, with several additional rows for about 150 ft. at the west end. On May 13, 1930, these holes were blasted, breaking 275,000 tons—the largest blast ever made at the Tilden. Another was made on Nov. 1, 1930, at the close of the season, when one row along the entire face of the pit, together with a field blast at the west end, was fired. Table I presents data regarding blasts made in the season of 1929. Table II gives corresponding data for that of 1930.

Two blasts were made in 1930, one on May 13 and one on Nov. 1. All ore broken by the former was loaded that year, whereas that broken on Nov. 1 counted for the season 1931, and was blasted only to make drilling possible through the winter of 1930-31. The amount of powder and fuse consumed



Loading ore in the Tilden open pit

After the blast of a double row of holes on May 13, 1930. The ore is ready for loading

Table I—Blasts at the Tilden Mine, 1929

Blast No.	Date	Section of Pit	No. Holes	Total Depth	Pounds Powder	Est. Tons Broken	Tons per Ft. of Hole	Tons per Lb. Powder
1	April 16.....	East.....	85	3,610	20,000	80,000	22	4.0
2	May 15.....	East.....	20	1,960	18,800	50,000	25	2.6
3	June 18.....	Center.....	13	807	8,400	25,000	30	3.0
4	July 13.....	Center, W.....	47	2,333	20,650	50,000	22	2.4
5	July 24.....	East.....	21	1,647	12,150	38,000	23	3.1
6	Aug. 14.....	Center, E.....	4	325	2,300	7,000	21	3.0
7	Aug. 21.....	East.....	17	1,425	13,600	40,000	28	3.0
8	Sept. 7.....	Center and West..	57	3,400	28,300	95,000	28	3.3
9	Oct. 4.....	Center.....	17	1,610	12,000	40,000	25	3.3
10	Oct. 14.....	West.....	12	470	3,400	10,000	21	3.0
11	Oct. 26.....	West.....	21	1,450	12,500	40,000	28	3.2
	Totals.....		323	19,037	152,100	475,000	25	3.12

Table II—Blasts at the Tilden Mine, 1930

Blast No.	Date	Section of Pit	No. Holes	Total Depth	Pounds Powder	Est. Tons Broken	Tons per Ft. of Hole	Tons per Lb. Powder
1	May 13.....	Ent. Face.....	131	10,951	80,650	275,000	25	3.40
2	Nov. 1.....	Ent. Face.....	94	7,567	48,900	175,000	24.4	3.57
	Totals.....		225	18,518	129,550	450,000	24.3	3.48

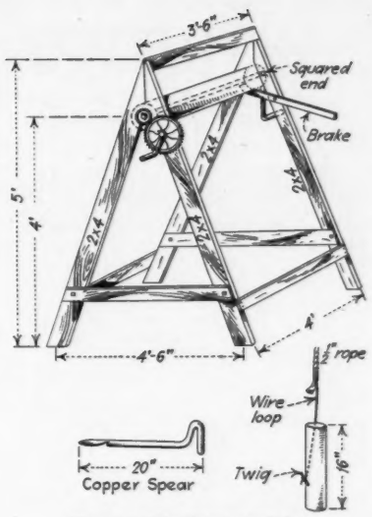


Fig. 3—Windlass used for lowering cartridges in blast holes in Tilden mine

in these two blasts is given in Table III.

The 6-in. blast holes are drilled with electrically driven churn drills. Five No. 14 Sanderson Cyclone drills were used until June, 1930, when an Armstrong No. 29, an all-steel drill, was added.

The rock in the Tilden pit is very hard and abrasive, and is badly fractured in places. Drilling blast holes presents a problem, and by continually experimenting, improvements in operations are being made. Though a maximum of 35 ft. has been drilled in a ten-hour shift, average footage per shift is about 18.

The bits used are formed in a No. 6 Armstrong bit-dressing machine, and sharpened with a 6-in. Type D, 8-deg. taper die. The hard and fissured ground causes much trouble, in that holes go off line and drill bits lose gage. The Type D bit has the advantage of an outside reaming edge, which seems to keep the hole in line. Drill runners are taught to watch the gage so as not to wear it down too far, thus making it impossible to get a sharp bit down the hole. In certain spots the ore is so hard that a properly tempered churn-drill bit is dulled after drilling from 1 to 2 ft. of hole, although the average is about 7½ ft. per bit. To avoid so far as possible having a new bit stick in the hole on account of wearing the gage of the previous bit down too far, the bottoms of holes are sprung by blasting with two sticks of 1¼-in., 60 per cent gelatin powder. Fig. 6 shows a sketch of the bit used.

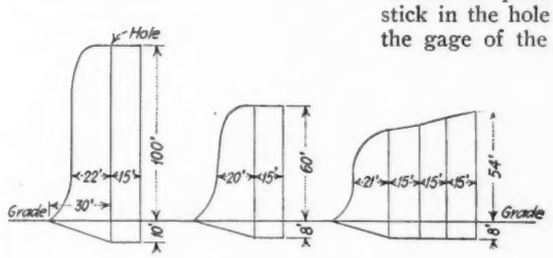


Fig. 2 — Typical drill-hole sections at Tilden mine

Churn-drill holes are spaced 15 ft. apart along the face, and 20 ft. behind the row previously blasted. When more than one row is drilled, rear holes are staggered on a spacing of 15 by 15 ft. maximum. Before a blast, the next row of holes is laid out by painting marks on the ledge to insure proper spacing. Holes are drilled 8 to 10 ft. below grade, depending on the dip of the formation, which varies 50 to 80 deg., and is to the north or into the hillside; also depending on the way in which the previous blast broke the toe of the cut. Except when a hole has been lost close to the finished grade, holes are not chambered.

The amount of powder required for each hole is calculated from survey measurements taken in the field just before each blast. Hole, crest, face, and toe are plotted as a section to scale, and the area of the section is multiplied by 15, the spacing of the holes. Fig. 2 shows several typical drill-hole sections. This gives the burden in cubic feet, which divided by 45 cu.ft. for front holes, and 35 cu.ft. for field-blast holes, gives the pounds of powder necessary to the burden of the hole. The factors of 45 and 35 cu.ft. were determined by experience. The amount of powder used is proportioned by scaling the distance between the pit face and the hole at different depths. For distances greater than 25 ft., Gelatin Extra L.F. of 90 or 80 per cent strength is used, and 60 per cent Gelatin L.F. and Gelamite 2 are used in proportions necessary to complete the charge.

Experience shows that 100 lb. of gelatin in 5x16-in. cartridges takes up 7 ft. of drill hole; so that, if the burden on a hole is greater than 25 ft. for the bottom 20 ft. of hole, about 100 lb. of 90 per cent strength and 200 lb. of 80 per cent strength would be the desirable charge. In deeper holes, additional amounts of Gelamite 2 are often deck-loaded to shatter the ore in the upper section. The powder charge is brought within 18 to 20 ft. of the collar of the hole by deck-loading. Some trouble was experienced in earlier blasts in breaking the lower seams to grade, but the use of 90 per cent dynamite instead of a closer spacing of drill holes corrected this difficulty. From 10 to 15 ft. of damp sand or drill sludge is used for

Table III—Powder and Fuse Consumed, Blasts of May 13 and Nov. 1, Tilden Mine

	Blast of May 13	Blast of Nov. 1
Cordeau-Bickford Fuse, ft..	13,000	10,139
40% Hercules Powder, lb....	1,450	.....
60% Hercules, Powder, lb...	6,900	.....
40% Gelatin, lb.....	.....	800
60% Gelatin, lb.....	300	5,200
80% Gelatin, lb.....	32,000	20,550
90% Gelatin, lb.....	2,500	2,550
Gelamite No. 2, lb.....	37,500	19,800
Total powder, lb.....	80,650	48,900
Estimated tons broken...	275,000	175,000

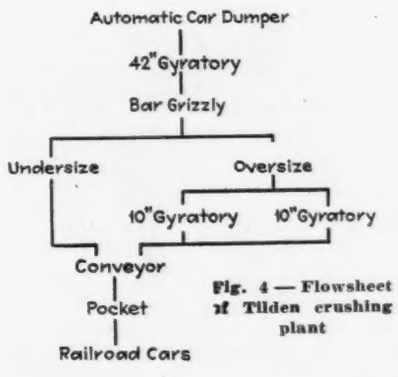


Fig. 4 — Flowsheet of Tilden crushing plant

stemming material in all the dry holes.

In loading the holes we use crews of two men to a hole, with several extra men to carry powder to the loaders. The usual rules, of "no smoking" and keeping everyone away from the loading operations except those directly concerned, are strictly enforced. Before loading is started, the rammer is first let down the hole to check depth and deep holes is accelerated materially by using the windlass. No time is lost in untangling the rope, as when lowering by hand. A loop of copper wire, attached to the end of the half-inch rope on the windlass, is pushed through the cartridge with a copper spear, being inserted so that it will come through the side about three-quarters of the distance down. A small thin piece of pine is then put through the loop to keep it from

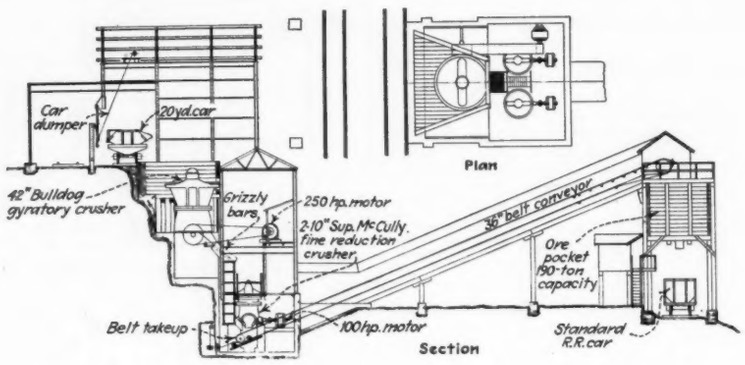


Fig. 5—Longitudinal section of crushing plant. Plan view of crushers.

condition. If the hole is ready, the Cordeau primer is prepared by lacing the Cordeau through the cartridge, going first into a hole diagonally from the center of the top to the outside about 10 in. down, and then wrapping the Cordeau around the cartridge and bringing the end through the loop, thus forming a half hitch. Double-countered Cordeau is used in shallow holes; and in deep ones, where abrasion of powder in lowering it down the hole is great, wire-countered Cordeau is used. Plain Cordeau is used for the trunk line in connecting holes.

Powder cartridges are lowered into holes instead of dropping them, so as to provide a constant check on the rate at which the powder rises in the hole. Mr. C. W. Allen mining engineer at the Tilden mine, in charge of loading operations, designed a windlass, shown in Fig. 3, for lowering powder. The construction and operation of this device was described in the August, 1930, issue of *Explosives Engineer*. Loading of slipping back. When the powder reaches the bottom of the hole it is jerked up and down a foot or so until the pine stick breaks and allows the wire to pull out of the powder. A wooden dolly, weighing 32 lb., attached to a rope, is used for tamping powder.

After holes have been loaded, the trunk line of plain Cordeau is strung and the mining engineer and blasting foreman split stems and make connections, which are inspected carefully by the engineer in charge and the superintendent. If inspection is satisfactory, the lead wire is reeled out to the blasting station in a sheltered house some distance away. Two electric blasting caps are inserted in one end of a stick of 1¼-in. 60 per cent Gelatin powder and the trunk line Cordeau is inserted in the other end. Two blasting caps are used to insure the blast being set off. All watches are synchronized, and a definite time is set for blasting. Men are sent to guard all highway approaches and railroads, and the blast alarm is given by the locomotives, after which the blasting foreman connects the lead wire to the blasting battery, which has previously been tested, and sets off the blast at the appointed time.

Broken ore is loaded by two No. 80-B Bucyrus-Erie electric shovels with a 2½-yd. dipper, and manganese-steel teeth of the Van Port reversible type. Average life of a set of teeth is between 15,000 and 20,000 tons. These shovels are shown in accompanying illustrations. Conditions required one of them to be moved along the Cliffs Drive road to the Tilden from the Ogden pit, in the fall of 1928. It negotiated the five miles under its own power, made possible by stringing a transmission line close to the road on trees and poles. From this, connections were made with the shovel cable at intervals of 500 ft.

The ore is transferred from the pit to the crushing plant by three Pittsburgh standard-gage, 17x24-in., four-driver, 40-ton locomotives. Each hauls a train of four Phoenix duplex-type cars, manufactured by the Easton Car & Construction Company, of Easton, Pa. Each car is built with two bodies having a capacity of 10 cu.yd.

Pit cars are dumped directly into the hopper of a 42-in. Traylor Bull Dog gyratory crusher, set to a 5-in. opening, and driven by a 275-hp. motor. Any chunk that will pass through the dipper of the shovel will go through the crusher without difficulty. The car bodies are dumped by an arm that engages a rod on the side and is lifted by an electric hoist. The hoist is equipped with a solenoid brake, so that the car can be held at different inclinations, thus controlling the speed with which the ore is fed into the large crusher. After the car body is emptied, by means of a counterweight attached to the dumping arm, it is lowered to the chassis, the dumping arm swinging out of the way to await the next car.

The ore from the large crusher passes over a manganese-steel grizzly. Oversize goes to two 10-in. Superior fine-reduction crushers, each set to a 2-in. opening, and driven by 100-hp. Allis-Chalmers motors. Undersize from the grizzly and the product from the two 10-in. crushers are fed on to a 36-in. conveyor belt and carried up a 22-deg. incline to the railroad pocket. The conveyor belt is of the Boston Woven Hose duplex type. The flow sheet of the crushing plant is given in Fig. 4. Plant capacity is 300 tons per hour.

Office, warehouse, substation, compressor room, machine and blacksmith shop and change-house are all under one roof. The compressor furnishes air for blockholing the large chunks and for drilling high spots in the pit bottom. The blacksmith shop is equipped with a No. 6 Armstrong bit-dressing machine, used for dressing the 6-in. churn-drill bits, and also a Sullivan drill sharpener for sharpening the drills used in blockholing.

Fig. 6—Sketch of churn-drill bit

Potash Mining in Germany

Ernest R. Lilley

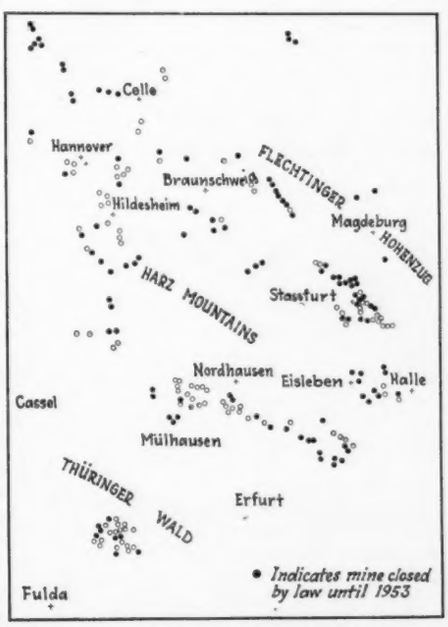
Mechanical conveyors moving blasted material to a main haulage loading point

THE potash deposits of Germany were discovered in 1851 by the Prussian State Government while testing for rock salt near Stassfurt. In common with many other outstanding discoveries, the value of the rich bed of potash lying between the beds of common salt was not immediately seen. Full appreciation came in 1857 with the announcement of the value of potash as a fertilizer. Mining was begun immediately and expanded steadily until the opening of the World War. Stassfurt was naturally the center of the earlier operations, but, as the diamond drill demonstrated the wide extent of the deposits, mining was carried as far south as Fulda and northward over the North German plain.

The demand from the German farmer, probably at that time the most progressive in the world, grew by leaps and bounds. As early as 1876, however, the larger producers of potassium chloride found it desirable to form a syndicate to regulate prices and control output. This and later syndicates gave a stability to the business that was sufficient in a rapidly expanding market to counterbalance the weakening influence of the extremely rapid increase in number of shafts and in productive capacity. By 1910, however, productive capacity had so far outstripped consumption that the Imperial German Government found it necessary to take a hand in regulating the affairs of the industry. With the loss of the Alsatian mines to France at the close of the War, Germany lost, also, its monopolistic control of the world's potash markets, and its greatly overexpanded industry faced disaster. To prevent this, the government further extended its control by a series of drastic laws that have had a material influence upon both economic and technical development in the German potash industry.

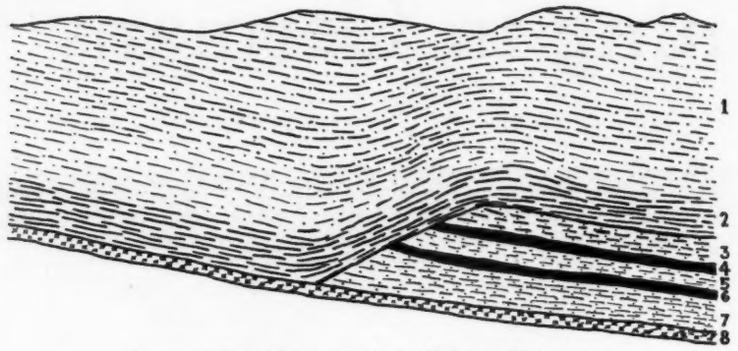
Though the United States is directly concerned in finding workable deposits of potash, the government of Germany is primarily interested in preventing the opening up of new deposits and in the closing down of excess capacity already developed. Tonnage estimates vary with the basis of estimation used and the personal views of the estimators, but all agree that Germany owns proved high-grade reserves in excess of all the other countries of the world combined, and could alone supply the world's needs for the next thousand years at the present rate of consumption without seriously depleting its potash resources.

By far the larger and better known of the potash-containing horizons is that of the Zechstein or Upper Permian. These beds were laid down in the relatively small basinal areas adjacent to the Harz and Thuringia Mountains and in the large Zechstein sea that formerly extended over the greater part of the area now occupied by the coastal plain of northwestern Germany. The deposits are in part highly folded and in part nearly horizontal in position. Depths vary widely not only between the different localities but also within the individual districts. Mining conditions and methods in consequence are far from uniform. Thus, although customarily in the past only the deposits in the immediate vicinity of Stassfurt have been thought of when the potash-mining industry of Germany was mentioned, and the inference made that geologic and mining conditions in that region were representative of the conditions prevailing throughout the whole of the German potash industry, this is no longer possible. In fact, the older mines in the Stassfurt region are today considerably less important than those in several of the more recently developed sections. German geologists and mining engineers now recognize seven distinct areas, each having its own peculiar geologic conditions and mining problems. These are: (1) the Northwest German plain or Hannover district; (2) the Stassfurt basin, lying between the Harz Mountains and the Flechtinger Hohenzug; (3) the South Harz basin, lying between the Harz Mountains and the Thuringer Wald; (4) the Halle-Mansfeld basin; (5) the Werra-Fulda basin; (6) the Lower Rhine district; (7) the Northeast Flank of the Flechtinger Hohenzug.



Sketch map showing Zechstein potash mines

The Zechstein beds lie immediately above a series of red sandstones and conglomerates that contain occasional thin seams of coal, known as the Rotliegendes series of the early Permian. Above the Zechstein lies a



Cross-section of a part of the Fulda potash area
(Adapted from Fulda)

1. Bundsandstein
2. Zechstein slates
3. Upper Older rock salt
4. Upper or Hessen potash bed
5. Middle Older rock salt
6. Lower or Thuringen potash bed
7. Lower Older rock salt
8. Middle and Lower Zechstein

complete series of Triassic rocks including the subdivisions known as the Bunter, Muschelkalk, and Keuper formations. These are partly marine and partly continental in character. The group contains several beds of highly porous jointed limestones that give much water trouble in shaft sinking and in later mining operations. These beds are in turn overlaid by Jurassic, Cretaceous, and Tertiary sediments. The younger formations are frequently very thin or entirely lacking, especially toward the south, but increase rapidly in thickness toward the north, so that in much of the North German plain the Zechstein formations can be reached at workable depths only in places where they have been subject to intense local uplift.

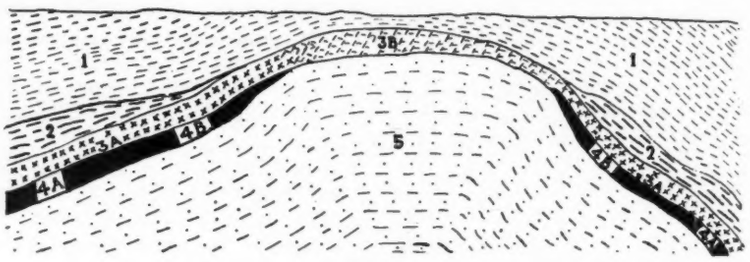
Comparatively little is known of the deposits in the Lower Rhine, as they have not been worked up to the present. Several mines were developed in the area north of the Flechtinger Hohenzug or uplift, but they did not prove especially successful from a financial standpoint and have been closed down. Mining is being actively carried on in all of the remaining districts. The best known to geologists is the area immediately around Stassfurt. Here the productive horizon seems to be mainly a bed yielding carnallite between 30 and 40 m. thick. A typical section of the Zechstein in this section, as given by Everding, is listed in the next column.

The succession is much the same south of the Harz Mountains, but the thickness of the Zechstein series is much less and the potash-containing layer is only 10 to 20 in. thick. In the Stassfurt district the potash-yielding horizon is present in the Hannover area. The "Younger" rock salt is substantially thicker and contains a second important and rich bed of potash, called the "Younger" or Sylvinite horizon. In the extreme southern portion of the Hannover district immediately west of the Harz Mountains, three distinct potash beds are recognized. In the Werra-Fulda district two main beds average from 10 to 15 m. in thickness each and are separated from each other by 50 to 75 m. of common salt.

Despite their geographic proximity, the potash deposits in Baden have an entirely different geologic background from those of the Zechstein. In fact, the Baden deposits are found in the Middle Oligocene and must therefore be grouped with those of Alsace geologically. The deposits were discovered near Muhlheim immediately prior to the War, but were brought to production only in 1927. This is a part of the Rhine Graben. The beds, which lie at a depth of approximately 2,600 ft., and consist of alternating series of salt and gypsum, appear to have been laid down in an area extending from the Vosges Mountains to the Black Forest, but commercial occurrence of potash is limited by the lenticularity of the potash beds. In the vicinity of Buggingen, the potash appears to be present over an area 10 miles long and from 4 to 5 miles wide. The potash is present as sylvinite, and the maximum thickness of the workable bed is only 4 m. The potassium oxide content, however, is high, averaging about 25 per cent. This is similar to the French and somewhat higher than the average of the Zechstein occurrences.

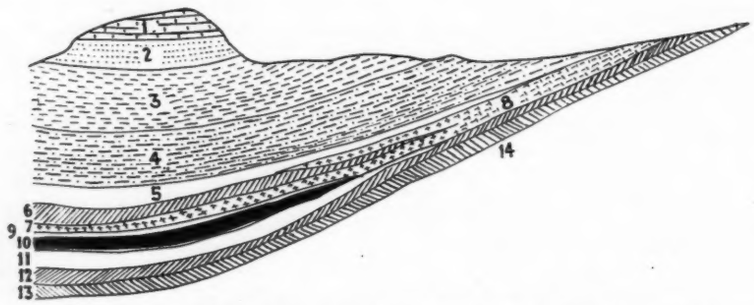
The attitude of the potash beds and of the formations associated with them varies widely in the several districts and is the cause of wide differences in mining practices and, in part, the ex-

	Meters
1. Compact red clay with lenses of anhydrite and laminations of rock salt with anhydrite at base	20–30
2. Rock salt	50
3. Anhydrite with crystals of rock salt	1–5
4. Compact red clay with lenses of anhydrite and laminations of rock salt	5–15
5. "Younger" rock salt	100–150
6. "Principal" anhydrite	40–90
7. Gray salt clay	4–10
8. Carnallite region	30–40
9. Kiserite region	20–40
10. Polyhalite region	40–60
11. "Older" rock salt	300–500
12. "Older" anhydrite region with intercalations of dolomitic bituminous clays and rock salts	70–100
13. "Zechstein limestone"	4-10
14. Cupriferous clay and the basal "Zechstein conglomerate"	½–4



Cross-section through a salt ridge in the Stassfurt district
(Adapted from Fulda)

1. Lower Bundsandstein
2. Younger rock salt
3. A. Anhydrite. B. Gypsum
4. A. Older potash as carnallite. B. Older potash as kainite
5. Older rock salt



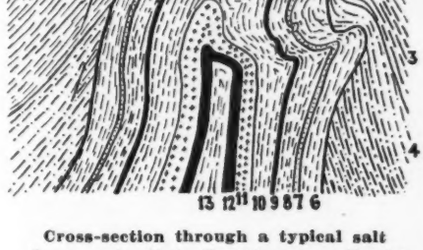
Cross-section of part of the potash area south of the Harz Mountains
(Adapted from Everding)

1. Lower Muschelkalk
2. Upper Bundsandstein
3. Middle Bundsandstein
4. Lower Bundsandstein
5. Secondary rock salt
6. Younger rock salt
7. Main anhydrite
8. Gypsum
9. Salt clays
10. Potash bed
11. Older rock salt
12. Middle Zechstein
13. Lower Zechstein
14. Rotliegendes and earlier formations

planation of the wide range in costs. In Baden, the potash horizon is conformable with the beds immediately above and below it, and, with them, lies in a nearly horizontal position. South of the Harz, the Zechstein beds are conformable with the horizons adjacent to them, but the whole group has been tilted substantially. In the Werra-Fulda district, the upper and lower potash beds are conformable with each other and with the older rock-salt beds in which they occur. They are separated by a distinct angular erosional unconformity, so that the potash beds terminate abruptly in the "Salzhang" with the Zechsteinletten.

North of the Harz, in the Stassfurt and the Hannover districts, the problems of the geologist and the engineer are not so simple. In contrast to the simple minor folding and tilting and the slight irregularities and variations in the thickness of potash beds found south of the Harz, the areas to the north are noted for their intense folding and wide variation in thickness. They constitute a veritable paradise to the student of salt mass tectonics. Thus, in the Stassfurt area, compressive movements from the Harz and the Flechtinger Hohenzug have resulted in the development of a series of elongated anticlinal ridges, in which the plastic common salt and the potash beds have undergone much distortion and vertical uplift. Not infrequently the potash beds lie at an angle of 45 deg. or more to the horizontal. Along the crests of the anticlinal ridges the anhydrite is commonly replaced by gypsum. In addition, the potash beds have been subjected to chemical alteration and, in large measure, removed from the crests of the folds by solutional activity.

In the Hannover district, where the thickness of the sediments laid down after the deposition of the Zechstein is great, the salt beds have been thrust upward for thousands of feet. The high plasticity of the potash and of the common salts has given rise to the most complicated systems of local folding in the salt masses themselves and, not infrequently, resulted in abrupt changes in the thickness of the highly desired potash beds. In some places the overlying formations have been lifted up bodily for hundreds of feet by the upthrust of the salt. In a large proportion, the strength of the overlying beds proved insufficient to withstand the upward movement of the salt masses, the salt breaking through and continuing its upward movement in much the same way as a highly viscous liquid might. Such occurrences present plenty of evidence that the uplifting and piercing of the overlying beds have been accompanied by intense faulting, most of it of the purely gravitational type. True faulting, however, has not apparently occurred in the plastic salt masses themselves.



Cross-section through a typical salt dome or horst in the Hannover district

1. Alluvial
2. Tertiary
3. Cretaceous
4. Jurassic
5. Gypsum cap rock
6. Upper Younger rock salt
7. Red salt clay
8. Middle Younger rock salt
9. Sylvinite region
10. Lower Younger rock salt
11. Main anhydrite
12. Carnallite region
13. Older rock salt

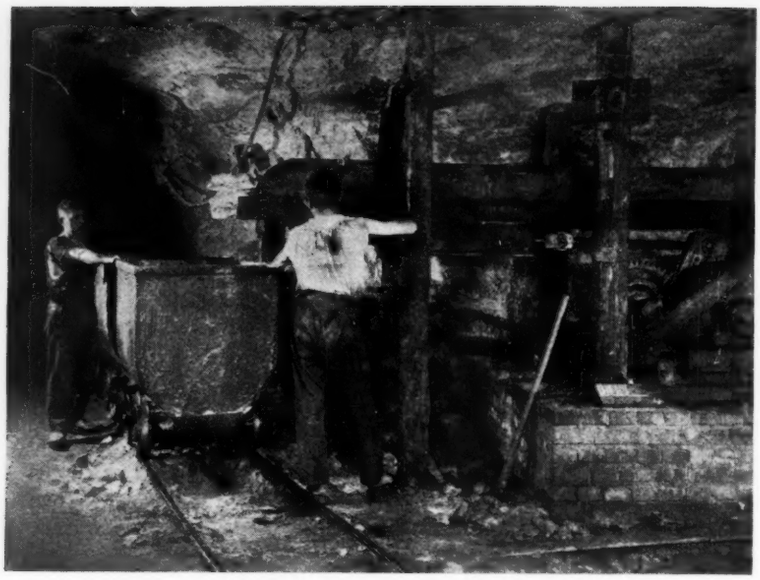
Drilling at the face prior to blasting

Although some of the salt masses of the North German plain appear to have the circular form and regular outline so characteristic of the salt domes of the Gulf Coast plain of the United States, the greater part are irregular in outline and are materially larger than those in Texas and Louisiana. Some are elliptical and others are distinctly elongated. Thus, one in the vicinity of Hannover has a breadth of 2 miles and a known length of 15 miles. The salt masses, "horsts" or "stocks," are usually capped by gypsum, probably derived in the main from the alteration of anhydrite. The upper portions of the potash beds may be entirely removed by solution. At present they are usually composed of secondary compounds derived from the alteration of older potash salts.

Although the original discovery of potash was purely accidental, practically all of the later discoveries have resulted from carefully organized exploratory work. Most of this work has been done with percussion and diamond drills. Percussion costs were uniformly lower than those obtained with diamond drills. The inability, however, to obtain cores and the uncertain and indefinite character of the well records obtained by the percussion driller gave the diamond drill a decided advantage. In general, the percussion unit seems to have been satisfactory for drilling through the overlying formations, except at points in which the sinking of shafts was contemplated. Here, and in drilling through potash-containing formations, the diamond drill has been used almost exclusively. The high solubility of the potash and associated salts introduced a problem not ordinarily present in diamond drilling—namely, solution of the core in the drilling water. This difficulty was met by substituting a saturated solution of magnesium chloride for the fresh drilling water commonly used.

Although Germany has been a leader in geophysical work, this method of exploration has been used but little by the potash industry, because of the legal restrictions placed upon the development of new mines since the War. Considerable experimental work has been done in the vicinity of the better-known salt horsts of the North German plain, and a number of new salt masses have been discovered by companies interested in petroleum development. If the need should arise, these new discoveries and many only partly outlined deposits could be further explored with success by geophysical methods. However, except as they are concerned in the possibility of oil development, the potash companies have no interest today in the discovery of new potash deposits.

The restriction against new developments has similarly retarded the use of the improved methods of sinking shafts developed by German engineers immediately prior to the War. A few of the earliest and shallowest of the German shafts were of rectangular cross-section,



Loading mine cars at discharge end of a conveyor line

but all of the later shafts have been of the circular type. In general, they average about 5 m. inside diameter. Although much of the distance from the surface to the potash horizons presents no particular problems, only a few of the earlier shafts were completed without long and costly struggles against inflowing water. Whereas some shafts have been completed to a depth of 600 m. in 1½ years, others have taken more than ten years to reach the same depth. Three special methods have been used successfully in sinking through water-bearing horizons where the older and more simple methods failed. These are the freezing process, the boring process, and the cementation process.

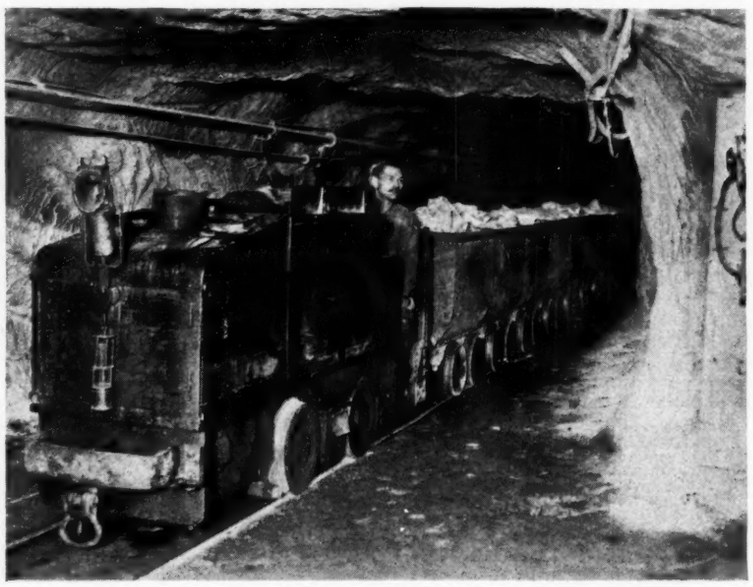
The freezing process has been favored where water-bearing sands and gravels of limited thickness were anticipated. Calcium chloride brine, and also alcohol, have been used successfully as freezing solutions, being circulated through tubes placed in holes drilled 10 to 25 m. in advance of the bottom of the shaft. Progress by this method was slow, much time being taken for the drilling of the holes in which the freezing solutions were circulated and in the freezing itself before any sinking could be undertaken.

The boring system, more commonly known as the Kind-Chaudron method, was designed to overcome the great loss of time where thick water-bearing horizons were expected. The sinking was carried on as two operations. First, a hole of from 1½ to 2 m. in diameter was made by using heavy drilling tools. This hole was then widened out by the use of specially designed boring tools to the full diameter of shaft desired. Cast-iron tubing was bolted together and lowered into the hole, in much the same way as casing is lowered into an oil well. The mechanical difficulties involved in sinking shafts in this manner increased rapidly as the diameter of the shaft was increased. Thus, most of the shafts sunk in this way were limited to an inside diameter of 4 m. The method seems, however, to have met with favor during the first part of the present century.

Early attempts to use the cementation process, now so widely employed throughout the Continent of Europe in coal shaft work, did not meet with success, lack of knowledge of the best methods of procedure resulting in failure of the cement to penetrate and seal up the water-containing pores properly. As improvements in technique were made, however, the process gained headway, and has been used with considerable success in a number of shafts completed during and immediately after the War. The value of this method has not been fully tested by the German potash industry, because of the restrictions on development work that came into force just as the possibilities began to be realized.

In the steeply dipping potash deposits of the Stassfurt and Hannover districts, some form of overhand stoping is invariably employed. Two distinctly different principles were formerly used. In the first, vertical and horizontal pillars were left to hold up the roof, and no attempt to backfill was made. The weakening of the salt pillars on exposure to the moist mine atmosphere led to their collapse and to the fracturing of overlying beds, with frequent sudden and disastrous inrushes of water. In the second system, no pillars were left, reliance for support of the hanging wall being placed entirely on backfilling, composed of the residual salt left after the removal of the potash in the leaching plant, and on common salt mined out of adjacent portions of the salt mass. Although fairly satisfactory in small workings, the compacting of the common salt backfilling under the pressure of the overlying strata was so great that subsidence, with all of its attendant ills, was most pronounced in the larger workings.

In present practice, the two systems are combined. Rooms varying up to 100 m. in length and 20 m. in breadth, depending on the thickness of the potash and general working conditions, are opened up parallel to the strike. If the beds are wider than 20 m., rooms are made perpendicular to the strike. Such rooms are separated from each other by pillars 5 to 6 m. thick. The natural strength of the common salt and potash in the roof, walls, and pillars is sufficient to hold up the rooms until a height of about 12 m. has been obtained. Then backfilling is begun. Overhand stoping and backfilling proceed simultaneously, until the potash has been removed throughout the stope. At no time is the backfilling allowed to lag far behind the removal of the potash salts, so that when the individual stope has been worked out, usually to a vertical



Trainload of potash salts en route to the hoisting shaft.

height of about 80 m., all of the potash extracted has been replaced by common salt. This combination system has been employed successfully and has resulted in a material reduction of the number of roof failures and water bursts.

A modified form of the same system is used with beds of moderate dips. In nearly horizontal deposits, room-and-pillar systems have been widely used. Rooms vary from 9 to 20 m. in width and up to 200 m. in length. Length of the room parallels the strike in some mines, and in others the dip. Usually the room is opened up for its full length and breadth to a height of 2 m., the broken salt being removed as it is blasted down. Overhand stoping is then begun, and the broken rock allowed to accumulate to form the flooring, upon which the workers stand. When the barren salts overlying the potash are reached, the broken salt is removed, and the room refilled with common salt.

The comparatively soft character of the potash makes the drilling of blasting holes with electric or compressed-air hand drills a simple operation. Similarly, the amount of blasting powder used is low. Except for greater use of electricity, for underground haulage and hoisting, the German potash mines present no special points of difference from the practices in use in mines producing other materials under similar conditions. Although, in general, danger from inflammable gases in German potash mines is not great, several fatal explosions have occurred and have made suitable precautions necessary in development work. In a few instances, petroleum and petroleum gases have been encountered in the potash. These substances, however, are more likely to occur in the porous rocks above or below the salt than in it.

Little support is needed for haulageways and ventilation tunnels cut in common salt. Danger of subsidence in the worked-out sections, however, is always present and must be guarded against. In the first place, some of the mines are below towns and cities, where subsidence would involve heavy damage claims. In addition, subsidence frequently opens up crevices through which floods of water may enter the mine, causing immediate and complete abandonment of otherwise profitable properties. Flooding invariably means permanent abandonment, as the incoming water tends to dissolve and further weaken pillars and bring on wide collapse. Because of this, backfilling, either by hand or mechanical means or by hydraulic methods, must always follow closely upon the removal of the potash. With strong pillars and careful backfilling, and restriction of development and mining work, so as not to bring operations too close to highly porous and fractured water-saturated rocks surrounding the salt, danger resulting from water can be avoided.

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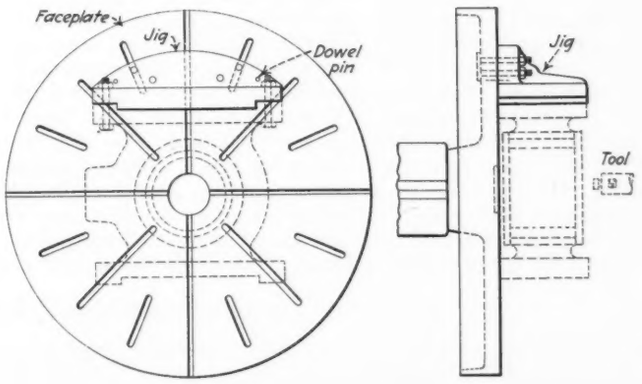
Jig for Diesel-Engine Crosshead Bearings



TURNING of crosshead bearings in a lathe is usually a difficult job, unless some kind of a jig or fixture is used, writes F. R. Kennedy, master mechanic, Old Dominion Company, Globe, Ariz. The bore and bearing centers must be uniform, to prevent undue wear on the guides and shoes. In the illustration is shown a simple method for machining crosshead bearings quickly and accurately. The bracket-like jig, held to the face plate by four ¾-in. bolts and two guide dowels, has a recessed face, carefully machined to accommodate and align the bolted-on bearing.

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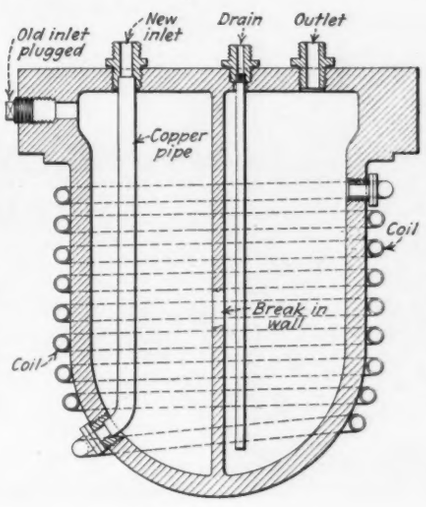
Novel Aftercooler Repair



THE aftercooler of an air compressor after many years of satisfactory service, suddenly failed to cool the generated air to the required temperature, despite an ample supply of cooling water and a frequent cleaning of the pipe system, writes Joe Carothers, mine master mechanic of Old Dominion Company, Globe, Ariz. An inspection of the unit during a brief shutdown revealed a break in the thin wall between the receiving and the discharge chamber, permitting most of the hot air entering the receiving chamber at the side of the cooler to pass directly into the discharge chamber, instead of first flowing through the cooled copper coil. With no spare at hand, the damaged aftercooler head was again rendered serviceable, at a low cost, by plugging the inlet on the side of the cooler and providing a new inlet on top of the cooler. The new inlet was connected to the coil end by a copper pipe, as is shown in detail in the illustration at the immediate left.

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Glaciation, Depth of Frost, and Ice Veins of Keno Hill and Vicinity, Yukon Territory

Livingston Wernecke

A CLEAR UNDERSTANDING of the surface conditions at the Wernecke camp of the Treadwell Yukon Company, Ltd., 40 miles north of Mayo and 100 miles east of Dawson, near 64 deg. north latitude and 135 deg. west longitude, can be obtained by a study of the illustration in Fig. 1. The photograph here reproduced was taken from the westerly end of Keno Hill at an altitude of 4,800 ft. looking S. 75 deg. W. down the McQuesten valley. Lookout Mountain, in the distance, is a monadnock, 6,600 ft. high, in a glaciated area—an excellent beacon for aviators coming to Mayo.

The first discovery of silver ore on Keno Hill was made by Louis Bouvette in 1919. The first discovery of silver ore in the district was in 1912 by Henry McWhorter, on Galena Creek, on the Silver King claim. Ore was discovered on what is now the property of the Treadwell Yukon Company by H. A. Stewart, associated with D. A. Cunningham, in the fall of 1920, on the Ladue claim about 75 ft. down the slope from the abandoned No. 1 shaft (1) in the halftone, near the lower edge of the concentrate platform; but, as winter was rapidly approaching, the discoverers did no further work until June or July of the following year. The discovery consisted of several pieces of gray carbonate-coated galena float. The Treadwell Yukon company became interested in the property in August, 1921, after the locators had uncovered a vein 4 to 5 ft. wide in a trench 5 ft. deep and 30 ft. long. Before the company acquired options on this claim, Keno Hill, Ltd., had acquired options on the adjoining Sadie and Friendship claims, where a second discovery had been made, which later proved to be the same vein. The Keno Hill shaft on the Sadie claim is shown at (3). Between that and the No. 2 shaft (2), on the Ladue, the Friendship and Bluestone claims are located, neither of which contained any strong orebodies, and only enough ore to pay the purchase price above the operating costs. The Sadie claim was later operated by the Treadwell Yukon Company, Ltd., under a lease from the Keno Hill company.

Lucky Queen mine of the Treadwell Yukon company, is above the camera to the left, not visible in the panoramic view. Two towers of the rope tramway leading to the Lucky Queen can be seen in the foreground on the left

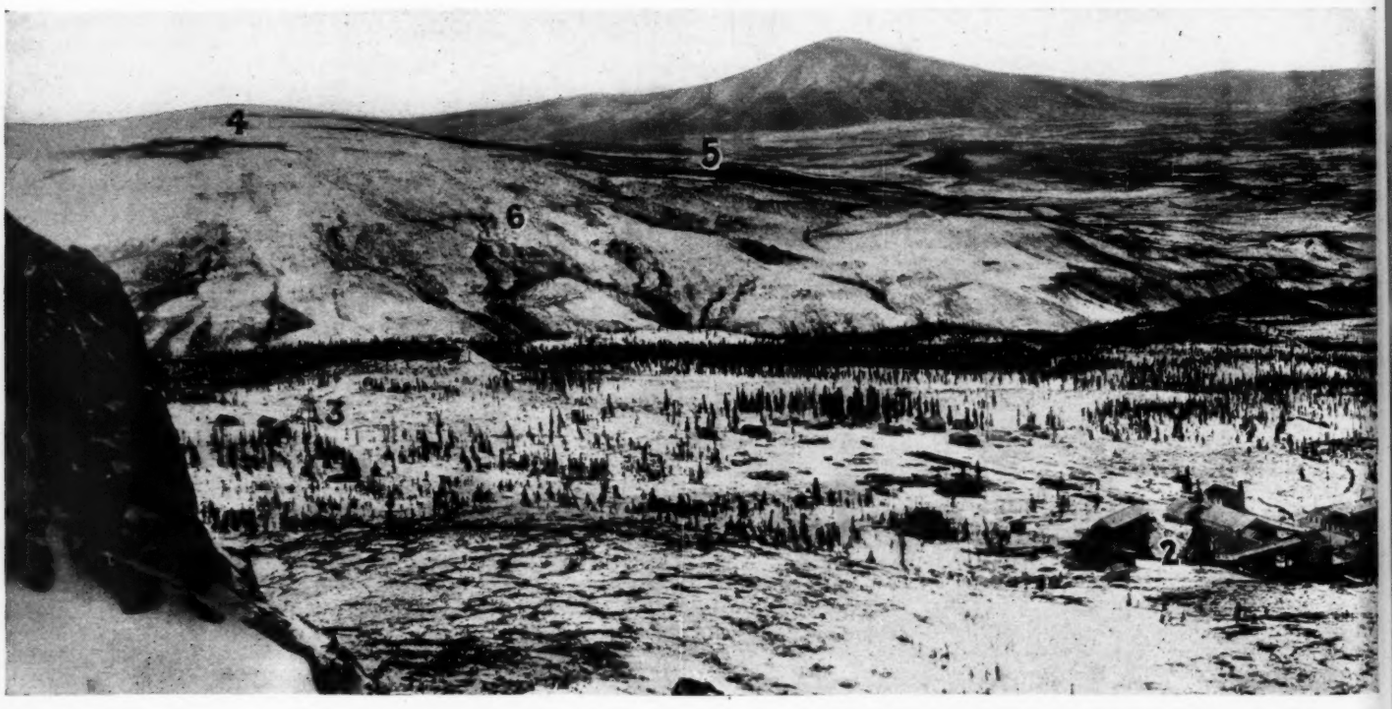
In the picture, Galena Hill (4) is at the left across Crystal Creek; and, like Keno Hill, contains many silver-lead prospects that cannot be operated at present metal prices.

Roads leading to and the clearing near the Silver King mine can be seen at (5), 10 miles distant.

The first silver-lead ore shipped from the Mayo district was mined there in 1914 by Jack Alverson and Grant Huffman. From 1929 until the recent decline in silver prices, the Treadwell Yukon company was developing this ground, where a new body of ore had been discovered by C. A. MacPherson.

Three miles toward the camera—again referring to the views at the bottom of this and the opposite page—from the Silver King is the Elsa mine behind the spur of Galena Hill and not visible in the picture. Here the Treadwell Yukon company has developed a small body of high-grade silver ore that can be shipped even at present low metal prices. Besides the high-grade ore (400

Fig. 1—Treadwell Yukon Property



Observations of more than ordinary interest will be found in this article. Although the frozen zone at Keno Hill persists in places to depths as great as several hundred feet, the ground temperatures there are said to be only a little below freezing, and to increase about 1⅓ deg. F. for every one hundred feet. Many who are unfamiliar with the North will marvel a little; first, because the ground is so cold; and, next, because it is not colder. Water trouble begins below the frozen zone, when the latter is pierced. The crystallizing force of the ice has evidently been responsible for the filling of the fractures, or their enlargement to present size. When the vein ice melts, the walls of the fractures collapse, proving they could hardly have been standing open prior to the filling. Within the ice, whether clear or opaque, rock fragments, evidently "plucked" from the walls by the formation of ice crystals, can be seen. These occurrences, Mr. Wernecke believes, offer support for the theory that some types of metalliferous veins are formed by growing crystals. And, reasoning from his observation of ice phenomena, he adds that an apparent replacement may be attributable to the separation of the "unreplaced" particles by the crystal growth of quartz. These conclusions, together with Mr. Wernecke's denial of the accepted belief that the upper remnants of the Yukon plateau are unglaciated, should stimulate interest and encourage discussion.

oz. silver per ton), additional ore of a milling grade has been developed, which must await improved metal prices.

Of interest to the geologist is the uniform bench line around Galena Hill (6), which represents a lateral moraine deposited during the period when the McQuesten valley glacier was at its maximum. On Keno Hill, below camp, this lateral moraine is at 4,000 ft. elevation, and it decreases in altitude down the valley. Although no striking evidence is available locally to prove that glaciation extended above the lateral moraine shown in the photographic reproduction, proof exists elsewhere in the Yukon of at least two periods of glaciation, and a few indications suggest a general inactive snow cap for the Keno Hill district that covered many of the surrounding mountains and hills during the first and larger glacial period.

In some places in the Yukon Territory, such as on the headwater of the Pelly River drainage, the general snow cap became an enormous active glacier, which moved westward down the Pelly valley, overriding most of the high lands and filling many pre-existing valleys. It drained across the direction of the movement, with morainal material. This glacier left deep scorings, also, on the hilltops, which might be likened to *roches moutonnées* on a large scale. The general glacier was followed at some later time by valley glaciers, which occupied the trenches of the present Pelly River and its tributaries, and which left ground moraines, lateral moraines, and recessional moraines with the knob and basin topography similar to that in McQuesten valley. Some of the valley glaciers (those tributary to the north side of the Pelly River) crossed the older scorings and *roches moutonnées* at almost right angles, thus establishing at least two periods of glaciation for the upper Yukon valley.

It was the ice-cap glacier flowing northwestward that filled the Tertiary Livingston Creek gulch in the Big Salmon district with morainal material and preserved its placer gold until the stampede of 1903. Possibly the gold in the tributaries to Pine Creek of the Atlin district may have had the same history.

The snow cap about Keno Hill left a few small isolated foreign erratics, unmixed with fine gravel or clay, on the flat-topped hills, and with a thin layer of blue clay in the more protected spots. The isolated erratics are well rounded and water-worn. They may have been derived from some Tertiary stream bed that existed prior to the elevation of the Yukon plateau in the Miocene or Pliocene. Space is not available for describing the Tertiary peneplain that has been elevated to form the Yukon plateau, and its subsequent dissection by rivers and glaciers. The general snow cap, having left no prominent glacial scorings, striations, moraines, or till on the hills or mountains in the immediate vicinity of Keno Hill, is thought to have been mostly inactive, owing to a semi-arid climate when the annual evaporation about equaled the annual precipitation[1]. The active ice of this vicinity was confined to the valleys.

A clever bit of reasoning was done by "Sandy" (C. A.) MacPherson con-

at Camp Wernecke, Yukon Territory

Fig. 2—Ice veinlets in slate at face of 45-deg. raise, Ladue mine, about 75 ft. below surface, having all the characteristics of some irregular quartz veinlets. Some small inclusions of wall-rock are visible.

Fig. 3—Ice veinlets along set of fractures in slate and quartzite. Face of crosscut on 50-ft. level, Ladue mine.

cerning the effect of this valley glacier on the float from the outcrop of the Silver King vein near the foot of Galena Hill. The vein strikes diagonally downhill southwesterly, so the ice moving along McQuesten valley should push the float ore and vein outcrop material above the apex of that part of the vein that is further southwest or downhill along the outcrop. To trace the vein from such float, MacPherson reasoned that he should trench and follow the float downhill instead of the usual method of following float uphill. This he did, and he located, in 1929, an extension of the Silver King vein, for which others had been seeking since the first discovery in 1912.

On the right center of the panoramic view, extending entirely across McQuesten valley, is a wide crescent-shaped group of knobs and basins, which is a recessional moraine of the McQuesten glacier, representing an extended halt in the gradual retreat of the glacier, rather than any actual advance of the ice tongue.

[1]I am mindful of the fact that many members of both the Canadian Geological Survey and U. S. Geological Survey, who have seen and described the Yukon plateau, have the opinion that its higher remnants are non-glaciated areas, since the ice of the glacial period occupied the older river valleys. I also held this opinion until I had the opportunity to see the topography toward the head of the Pelly valley and to consider the fact that no soil commensurate with a surface exposure since mid-Tertiary is found anywhere on the higher plateau remnants. The surface of these high plateau remnants, tablelike lands, is covered with 5 to 10 ft. of angular frost-fractured rock fragments, varying from the finer material to 12- and 18-in. diameter. Only a very small amount of soil from rock decay or weathering can be found. This suggests that, at some time, the Yukon plateau was swept clean of its Tertiary soil, which was a product of rock disintegration in milder climates.

Hansen Lake (7) and the other smaller lakes across the valley are marginal lakes left by the retreat of the glacier from its recessional moraine. In them, varved clays may have formed from which someone may some day determine the time since the glacier disappeared.

Frozen Zone—In the left foreground of the panorama are some miniature roundish humps separated by small connected depressions in which small shrubs are growing. These humps are the result of freezing and thawing of water in mixed soil and broken rock, a process similar to convection in water prior to boiling, but differing therefrom in that the process is extremely slow. This transportation of rock fragments from the bottom to the top serves to identify the covered bedrock and also exposes vein filling, if veins are present. Frost effect in high latitudes has been described by H. M. Eakin (Washington Academy of Science, 1914), who has designated the process mentioned in the foregoing as "altiplanation."

The thickness of the frozen layer of soil and solid rock in this vicinity varies with the elevation, hillside exposure, depth of clay, gravel and tundra overburden, and nearness to running water. At the head of Faro Gulch near the top of Keno Hill, at an elevation of 5,600 ft., where there is a cliff 900 ft. high on the north side, Keno Hill, Ltd., sank a shaft over 400 ft. deep in solid rock without reaching the bottom of the frozen zone. At the Lucky Queen mine, an elevation of 5,100 ft. and 4,500 ft. distant from the main camp shown in the picture, the shaft is 360 ft. deep and still in frozen ground. At the Elsa mine, 3 miles this side of the Silver King, a tunnel was driven 800 ft., until it had an overburden of 300 ft., where it encountered thawed ground. At the Silver King, the lower limit of the frost line is 200 ft. below the surface. At the No. 2 shaft of the Ladue mine, frost has penetrated the ground 260 ft. deep. At the Sadie shaft, the ground is thawed from the surface down. This thawed condition is caused by water running down a wide gulch to the left of the rock bluff on the left side of the view, sinking into the broken rock on the surface and eventually cutting through the entire frozen zone, which then permitted the underground water to come to the surface.

The temperature of the frozen ground is not extremely low, as shown by the following data for the Lucky Queen mine: 100 level (face 101 drift), —2¼ deg. C., 28 deg. F.; 200 level (face 204 drift), —1½ deg. C., 29.3 deg. F.; 300 level (face 302 drift), —¾ deg. C., 30.7 deg. F. This increase in temperature with depth is at the rate of ¾ deg. C. (1⅓ deg. F.) for every 100 ft. in the frozen zone. The rate of increase will probably be faster in the thawed ground. The thermometer is useful to indicate when the bottom of the frozen zone is near and water may be expected.

Underground Water—Wherever the frozen zone has been perforated, water under hydrostatic pressure has been encountered. When tapped in the No. 1 shaft, the flow of water flooded the lower level until adequate pumping facilities were provided. Eventually a drain tunnel was driven, which carried all the underground water from both the Ladue and Sadie mines in excess of that required for milling and domestic purposes; 100 g.p.m. flowed from the tunnel

of the Elsa mine when the frost line was first penetrated. At the Silver King, a winze that tapped "live" water was filled with concrete while ore from above the 180 level was being mined. The mine water in the Ladue mine contains 178 parts of solids per 1,000,000, most of which is calcium carbonate.

Springs flowing throughout the year and occurring high upon the slopes of the hills (mountains) are usually along some fault or vein system that extends through the frozen zone. These springs, particularly a line of them, are scrutinized carefully by the prospector for oxidized vein filling. Springs flowed from several points on a line between the Sadie shaft and the Ladue No. 2 shaft, which later became dry when underground exploration work connected the two mines.

Ice Veins—In the frozen zone, ice veins have been observed that have many features similar to mineral veins, except that the ice veins are always small—not over 6 in. wide. See Figs. 2, 3, 4, 5, and 6. However, a group of them in a sheeted zone may be 5 to 10 ft. wide. Although none of these veins has been exposed for its entire length, they are known to be short when compared with metalliferous veins. In stoping operations, one that occurred along a fault was exposed from the 200 level to a short distance above the 50-ft. level—about 150 ft. Those that fill irregular cracks are believed to be much shorter, with a maximum of 25 to 30 ft. The ice veins are larger and more abundant between the 100-ft. horizon and the surface than below that horizon. On the 300-ft. horizon, the ice seams are seldom more than ¼ in. wide, and bunches of ice of more than 2 in. diameter are rare.

The origin of these ice veins is easier to understand than that of many metalliferous veins. Some of the small bunches of ice are undoubtedly filling pre-existing irregular cavities, although these cavities may have been enlarged by the crystallization of that ice. These can be compared to the brecciated parts of mineral veins, where the spaces between wall-rock fragments are filled with minerals such as quartz, or siderite, or calcite. In other places, the ice occupies fractures and has an irregularly tabular shape as illustrated in Figs. 2, 3, 4, 5, and 6. This ice could not have filled pre-existing fractures having the same widths as now occupied by the ice. Three of the occurrences illustrated (Figs. 2, 3, and 4) are in friable slate, which will not maintain such openings for a sufficient interval to be filled with ice without sloughing full of fragments. The fourth (Figs. 5 and 6) incloses many blocky fragments in quartzite, similarly oriented. Each fragment is surrounded by ice. If these fractures existed previously to the ice, the fragments would have settled down and rested on each other. They would have a heterogeneous orientation. In another instance (not photographed) a 3- to 4-in. vein of ice was found along slickensides on the footwall of a fault which had a dip of 25 deg. Three or four inches of frozen gouge (a soft black mud when thawed) lay on the hanging-wall side of the *ice*. In this instance, the evidence is decisive concerning the formation of the ice vein by the crystallizing force of ice. No opening could have existed below the mud-like gouge when it was in a thawed condition, which must have been its state when water was in that vicinity and probably trickling along the footwall of the fault. As the temperature decreased, the water froze and a continued supply from below enabled it to pry apart the frozen mud and footwall rock by the force of crystaling water. No evidence has been noted that would hint the dissolving of the wall rock or gouge to make room for the ice vein.

The ice-limonite vein filling shown in Fig. 5 contains 58 per cent ice by weight, or 83 per cent by volume; the remainder is fine pulverulent hydrous oxide of iron and manganese, and a small quantity of antimony ochers. This material is from siderite oxidized prior to the ice age. Note arrangement of quartzite fragments parallel to walls, which is strong evidence of having been split from adjoining quartzite walls. Compare with photographic reproduction of argentiferous smaltite and calcite in diabase by W. L. Whitehead, *Econ. Geol.*, No. 2, of 1920, reproduced in Spurr's "The Ore Magmas," p. 134.

After the ground has been opened and warm summer air circulated through the mine workings, the ice of these veins melts, the walls of the fractures collapse if they are in slate; and, after two or three years, to find evidence of these veins is difficult, unless they are along a fault plane or prominent set of fractures, or in hard blocky ground. Even then the fractures are full of wall-rock fragments.

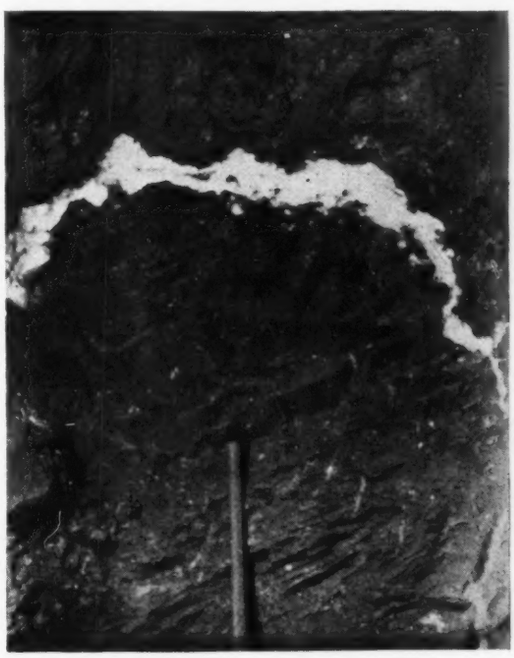


Fig. 4—Ice veinlet in face of adit level, Lucky Queen mine, about 75 ft. below surface. The vein dips toward the camera.

Fig. 8—Enlarged view, lower right-hand corner of Fig. 7; 2½ times natural size. Rough surface: limonite and manganese oxide; black smooth areas: clear ice in which can be seen small islands of limonite; white area at top: clear ice beginning to melt from light used in taking the photograph from which the cut was made.

Fig. 5—Sheeted zone, 6 ft. wide, of ice veins containing limonite, 100 level, Lucky Queen mine; only part of zone shows in photograph.

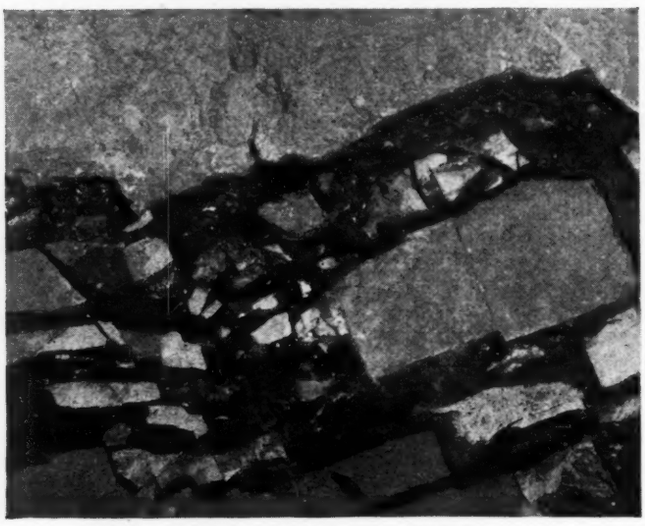


Fig. 6—Close-up view of part of Fig. 5 on left side, showing some of the small fragments pushed out of parallel alignment with adjoining walls.

Fig. 7—Ice-limonite vein filling from right-hand end of Fig. 6. White specks are ice reflecting light into lens of camera; black spots and streaks are clear ice also. One-half natural size.

All of this ice is believed to be ordinary ice "I" and not the high-pressure varieties, ice "II," "III," or "IV,"[2] all of which must have lower temperatures and extremely high pressures for their formation: —22 deg C. and 2,050 atmospheres for ice "III," and near 6,000 atmospheres for the others. The ice from the veins floats in water, which is a property of ordinary ice; whereas the high-pressure ices sink, being heavier than an equal volume of water.

Although the pressure actually exerted by the ice crystals in forcing the walls of the crevice apart cannot be determined, an approximate estimate can be made by assuming that the crystallizing force of water was expended toward lifting the superincumbent rocks—a static load; and, for a short interval, toward splitting the quartzite and slate. At a depth of 300 ft. the weight of the quartzite and slate is only 350 lb. per square inch, or about 24 atmospheres. If the cohesion of the rock molecules must also be broken to provide for the necessary 10 per cent increase in volume for the ice, the pressure at one time must have exceeded the tensile strength of quartzite, between 10 and 250 lb. per square inch (estimated, depending on the relative position of the fracture to the lamination or schistosity), plus the 350 lb. per square inch for the superincumbent load, or 360 to 700 lb. per square inch for the total pressure. These figures, approximations, represent only the order of the magnitude of the pressure that may have been exerted at one time by the ice crystals.

The water for these ice veins probably came from below. At the time of their formation, during the glacial period, there is reason to believe that surface temperature was constantly below the freezing point of water. Consequently, no water could have existed as a liquid at the surface. The underground supply was some meteoric water migrating along fault or fractured zones or vein systems toward regions of lower pressure.

All these ice veins contain some fragments of wall rock in the ice, unsupported by other fragments or the walls. The small pieces of rock or vein filling can be seen suspended in the clear ice. In the white opaque ice, the fragments can easily be dug out with a pick or knife; and they, too, are found to be entirely surrounded by ice. This was done in many instances. In some, the thickness of this ice was less than $\frac{1}{16}$ in., but in other places the ice was 1 in. or more thick between the fragment and its nearest support. The explanation of these unsupported fragments is the same as for the ice veins: the ice crystals have plucked fragments from the walls of the fracture, or from fragments resting on fragments within the

[2]Rastall: "Physico-Chemical Geology," Edward Arnold, London, 1927, p. 17.

fracture, and moved them away from the walls as the ice vein increased in width.

When the underground water froze, the calcium carbonate seems to have segregated and precipitated in spots as a soft, white, pulverulent (when dry) mass on the walls of the fractures. When wet, this substance is a soft, pasty mass. Some of this material was observed included as small irregular-shaped masses in the ice, but the greater portion was on the edges of the ice. The carbonate was pronounced in the ice in the siderite veins, and not evident in the ice surrounded by slate or quartzite. In the last instance, a very thin coating of limonite has precipitated on the slate or quartzite walls.

Strange to relate, a few of the bunches of ice in the Lucky Queen mine contain beautiful wires of native silver. However, the silver, which is supergene, is believed to have been formed prior to the ice age, and may owe its present position to the growing ice crystals, similar to the pebbles and bits of earth the frosts of early falls lift from the ground. The wires were frequently associated with the soft, white calcium carbonate. The quantity of this argentiferous ice is extremely small and of no economic importance.

The ice veins have demonstrated that they can form from the surface to a depth of 300 to 400 ft. in solid rock; that the force of crystallizing water can push apart the walls of a fracture at that depth; and that the ice can suspend fragments of the wall rock.

H. C. Boydell[3] has prepared a masterful "Discussion of Metasomatism and Linear Force of Growing Crystals," in which "the theory of vein formation by the direct action of the force of crystallization" is discussed and the conclusion is reached that a number of "weighty considerations" are opposed to the theory, among which is "Le Chatelier's theorem and exemplified by the effects of pressure on ice," but later, in a discussion with Doctor Taber, Doctor Boydell[4] explains that his thesis applied only to the "Linear Force of Growing Crystals and that volume pressure was not included in the 'Discussion.'" It is the increase or decrease in volume that is of importance to the mine geologist, and not so much "the linear growth of crystals." If Doctor Boydell's thesis does not apply to volume pressure of crystallization, the way is open for an undisputed formation of ice veins by the force of crystallizing water, according to George F. Becker's[5] conclusions in 1905 and Doctor Taber's conclusions[6] in 1918. Since it is an observed fact that these ice veins do exist under an environment which leaves no doubt that they have originated from the force of crystallizing water, and since they were able to form from the surface down to a depth of 300 ft. and more, I can see no reason why they could not have formed at greater depths if the temperature had been sufficiently low. These ice veins suggest in a miniature way that many of the epithermal and some of the mesothermal metalliferous veins of minerals such as quartz, calcite, or siderite may have developed in a similar manner, if allowance be made for additional features caused by more chemically active vein-filling solutions, and if due consideration be given to the fact that the ice is a crystallization of a one-component system (a simple physical phenomenon); whereas metalliferous veins are thought to have formed from a crystallization of several solutes from dilute aqueous solutions under varying temperature, pressure, and concentration —a most complex system for analysis by the phase rule. The ice veins lend support to the theory for the formation of some types of metalliferous veins by the force of growing crystals as proposed by Graton[7], Taber[8], Dunn[9], Stillwell[10], and others, but afford little or no support to "Spurr's vein-dike theory[11] when applied to the origin or "Lindgren's epithermal veins[12]."

A corollary to the formation of veins by the force of crystallization is suggested by Figs. 7 and 8, where the growing ice is seen to have separated small aggregates of a pulverulent limonite into forms resembling many of the criteria for replacement. If ice has done this by crystal growth, could not the particles and fragments of slate and schist often seen in quartz veins be separated, displaced, and have irregular indistinct margins by a similar crystal growth of quartz instead of the usually assigned process of dissolution of the surrounding rock, with or without chemical change, leaving ragged, indistinctly margined remnants of the country rock?

A close study of Fig. 8 reveals all the criteria for replacement, except the most important one—that of preserving the original texture of the palasome. Although the specimen has the appearance of replacement of limonite by ice, it really is a separation of the minute grains of limonite by the increasing volume of ice. If the force of crystallization can be invoked to account for certain veins, could not this same force be used to account for some of the phenomena now usually assigned to replacement, such as the irregular particles and fragments of slate or schist often seen in and around the margins of some quartz veins?

[3]Boydell, H. C.: "Metasomatism and 'Linear Force of Growing Crystals.'" *Econ. Geol.*, No. 1, 1926, pp. 1-55.

[4]Boydell, H. C.: *Econ. Geol.*, No. 2, 1928, p. 214.

[5]Becker, George F., and Day, Arthur L.: "The Linear Force of Growing Crystals." Proc. Wash. Acad. Sci., 1905.

[6]Taber, Stephen: "Mechanics of Vein Formation," *Trans.* A.I.M.E., Vol. 61, 1918, and "Discussion of Metasomatism and the Pressure of Growing Crystals," *Econ. Geol.*, No. 7, 1926, p. 717.

[7]Graton, L. C., and Lindgren, W.: Bull. 293, U. S. Geological Survey, 1906.

[8]Taber, Stephen: *Op. cit.*

[9]Dunn, E. J.: "Reports on the Bendigo Goldfield," Dept. Mines, Victoria, Australia, 1896.

[10]Stillwell, F. L.: *Econ. Geol.*, 1923, pp. 506-510, and 1929, p. 157.

[11]Spurr, J. E.: "The Ore Magmas," 1923, pp. 126-156.

[12]Lindgren, Waldemar: "Ore Deposits," 3d ed., pp. 516-597.

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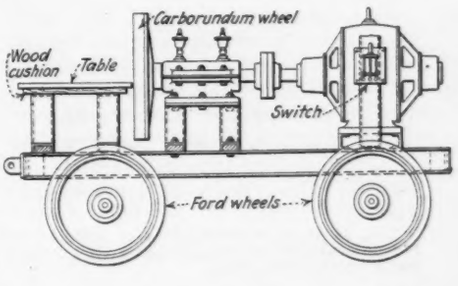
Portable Brick-Cleaning Machine

A PROBLEM confronting the maintenance engineer at the smelter is the large amount of old bricks available after a repair job on a furnace, a flue, or a boiler. Numerous bricks, perfectly sound except for the coating of mortar, are sent to the scrap dump, because the cost of removing the mortar exceeds that of the new product. To avoid such waste Charles R. Clarkson, master mechanic at the American Smelting & Refining's smelter at Hayden, Ariz., designed the portable cleaning machine shown in the accompanying sketch. It consists of a light truck of 4-in. channel iron, resting on four Ford automobile wheels; a grinding unit, with a carefully glued-on 24-in. diameter carborundum wheel, directly coupled to a 20-hp. motor; a table; and six channel-iron brackets. With a wheel speed of about 1,200 r.p.m., the capacity of this machine is over 500 bricks in eight hours, compared with 120 bricks cleaned by hand in the same period. When used extensively outdoors, the truck should be equipped with a light, removable cover, to protect the motor against moisture and dust, and a small top to provide shelter for the worker.



Economy Demonstrated by Underground Mechanical Shops

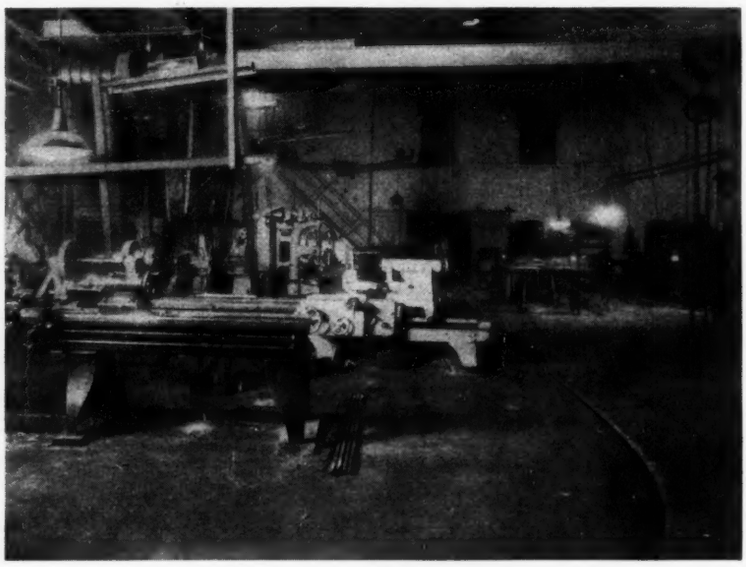
T. H. Garnett

THE shops at the Gilman, Colo., property of the Empire Zinc Company—subsidiary of the New Jersey Zinc Company—are situated in an underground room near the underground milling plant, which was described in an earlier article of this series. The room, 40 ft. wide, 80 ft. long, and 30 ft. high, was excavated by E. J. Longyear Company, along with the excavation of the openings for the milling plant. Like the other rooms, it is entirely in granite. A drift was driven to the west end and continued through to the east end. The sides of the drift were then drilled and blasted until the side walls of the shop were reached. Much was removed with Conway shovels. These shovels have a swing of 6 ft. on each side, so that most of the broken rock could be reached from three tracks on 12-ft. centers. The first cut was 9 to 10 ft. high. Rock above the cut was then blasted, and the roof trimmed from the top of the pile. Broken rock that could not be reached with the shovels was scraped to the center with a Lake Shore Engine Works double-drum scraper hoist, equipped with a 4-ft. scraper. It was then loaded into 50-cu.ft. gable-bottom, side-dump cars, with the power shovel. A 4x4-ft. raise, on a 70-deg. incline, was driven to the surface for shop ventilation.

The roof was gunited, using one part portland cement and four parts sand. The sand, obtained from the Fountain River at Pueblo, was screened through a $\frac{3}{8}$-in. screen, the larger sizes carrying the finer material through the hose and avoiding packing. Gunite was applied $\frac{1}{2}$ to 1 in. thick on flat surfaces and up to 6 in. thick in crevices. Staging, used throughout the room, consisted of 2x6-in. posts on 7-ft. centers, properly braced. A tight floor of 2x12-in. boards was laid on the staging, about 6 ft. below the roof. Carpenters were kept on the job to move staging and see that no boards became loose.

Two gunite machines were used, each working two shifts. Sand was dried outside the mine and mixed with cement in a small concrete mixer. The mixer discharged into cement sacks, which were filled only two-thirds full. This avoided tying of the sacks, and facilitated their handling. Each machine required a crew of one operator, one loader, and two men relaying on the nozzle. Three shifts of five men each were needed on the drying and mixing; another crew delivered mixed material to the gunite machines. The gunite has given excellent satisfaction. It prevents rock from coming loose, retards rock alteration, and improves the appearance of the shops.

After guniting, the sides and roof of the room were whitewashed, using a mixture of 5 lb. casein, 3 lb. trisodium phosphate, 50 lb. hydrated lime, 3 pints formaldehyde, and 25 gal. of water. This was mixed in an open barrel and poured into a pressure cylinder. A 50-ft. length of $\frac{1}{2}$-in. hose, fitted with a $\frac{3}{8}$-in. pipe nozzle, allowed the operator to spray all parts of the roof and sides without moving the tank. The end of the $\frac{3}{8}$-in. pipe was flattened to make a spray. Air pressure of 30 to 40 lb. was employed. The white walls make the



Main section of machine shop

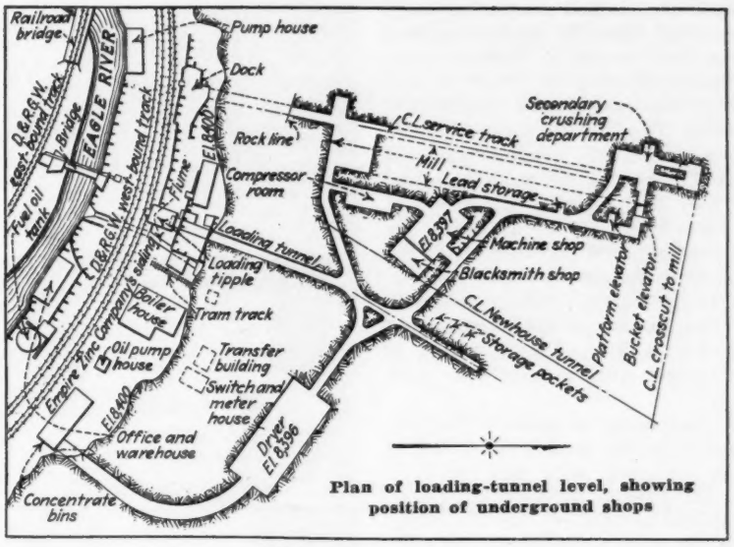
Blacksmith and drill-sharpening shops

This is another of a series of articles on the operations of The Empire Zinc Company at Gilman, Colo.

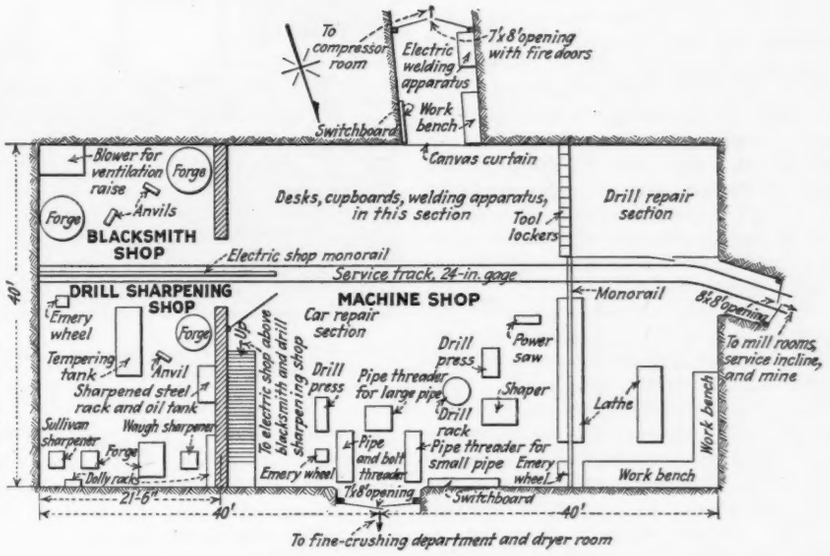
room much lighter and also add much to its appearance.

The shop is intended to serve both mine and mill, and is centrally situated among the mill openings. It is on the same floor level with the dryer, blower room, fine crusher, lower landing of the freight elevator, and lower end of the mill. Tracks of 24-in. gage connect all departments with the shop. The same gage is used throughout the mine. Drill steel or miscellaneous equipment from the mine is loaded on cars, lowered on the incline service track to the lower end of the mill, and then switched to the level track and taken to the shop. Storage-battery locomotives, used for handling concentrate, are also used in handling material and equipment to and from the shop. Supplies and light pieces of equipment for the mine are taken up the freight elevator and delivered to the dumping station on the Newhouse level.

After completing the excavation, the bottom of the room was leveled and a 4-in. concrete floor poured. A 24-in.-gage track runs the full length of the shop, 4 ft. south of the center line. Top of rail is flush with the concrete floor. The room has three entrances. The service track enters through an 8x8-ft. door at the west end. This track connects with all the mill rooms and, through the incline on the west side of the mill, with the Newhouse level. A 7x8-ft. door on the south side connects with the compressor room, and thence with the drift at the south end of the mill. A 7x8-ft. door on the north side connects with the drift that extends from the fine-crushing department to the dryer room. These are all double 4-ft. doors, set in concrete casing. Each door is made of a double thickness of 1-in. boards, the two layers nailed together, one vertical and the other at an angle of 45 deg.; all surfaces are covered with $\frac{1}{16}$-in. galvanized iron.



Plan of loading-tunnel level, showing position of underground shops



Plan of underground mechanical shops

In one part of the room is the machine shop, which includes a car-repair section, drill-repair section, tool lockers, electric welding, and steel fabricating. This shop occupies an area measuring 40x60 ft. A partition 20 ft. from the east end separates the blacksmith and drill-sharpening shops. The electrical shop is on a second floor above the blacksmith and drill-sharpening shops. This partition was made by laying up a 9-in. dry brick wall and guniting the sides. The gunite, driven in the spaces between the bricks, makes a tight and solid wall. Beams supporting the floor of the electrical shop rest on this wall and are supported by columns on the east end.

The machine shop equipment, in the northwest part of the room and occupying a 22x35-ft. area, includes a 16-in. lathe; a 34-in. lathe, with overhead I-beam and trolley from track to head of lathe; power hacksaw; 18-in. shaper; and a drill press. This equipment is all driven by a 7½-hp. motor, through an overhead line shaft, suspended from the roof and walls. The entire floor is entirely free from posts or obstructions. Additional equipment consists of a motor-driven grinder, small shear, and a 100-ton hydraulic wheelpress.

In front of the north shop door are two pipe-threading and cutting machines and a pipe-and-bolt machine. Long lengths of pipe can be handled by opening the doors. The larger machine handles pipe from 4 to 8 in. in diameter, and the smaller from 1 to 4 in. The pipe-and-bolt machine takes pipe from ½ to 2 in. and bolts up to 1½ in. Cars are repaired at the east end of the machine shop, where a drill press, emery wheel, work bench, cutting and welding equipment, and an overhead boom for lifting cars are available.

Rock drills are repaired in an inclosed corner, opposite the lathes. Equipment consists of work bench, vise, air connections, hand tools, and stock bins. This department also makes up and repairs air and water hose for the mine, and repairs small shop tools and equipment, such as oxygen and acetylene gages, power hand drills, and wrenches. The drift from the main shop to the blower room is used for electric welding. A set of standard fire doors on the blower side and a canvas curtain on the shop side shut off injurious light rays from other workmen.

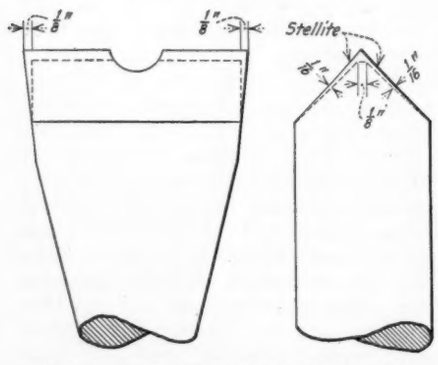
Welding is done with a 300-amp. General Electric arc welder. A work bench and tool drawer are available. The welder is equipped with mine car wheels and can be moved on tracks to service any point in the mill or mine. Long primary and secondary leads, and a few blank switches in the plant, make this unit particularly flexible. Space in the southeast corner of the machine shop is used for foreman's desk, tool and supply cabinets, and for steel fabricating.

Blacksmith and drill sharpening shops are separated from the main shop by a partition and occupy a 20x40-ft. area. The blacksmith shop is 15x20 ft. and is south of the service track. Equipment consists of two forges, two anvils, a vise, tool cupboards, work bench, and tool racks. Forges are coal-fired, and compressed air is used for blowing. Gases are removed by a 48-in. exhauster, which is on the floor above the shop and discharges through a raise, driven to the surface during construction. The suction of the exhauster is connected to a 30-in. header, which, in turn, connects with a hood over each forge.

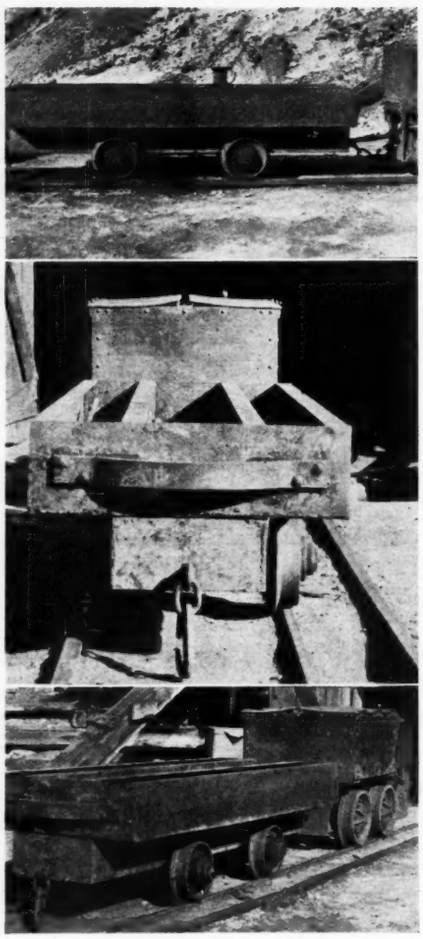
Drill sharpening occupies a 20x23-ft. area north of the service track. This shop is equipped with two oil-fired forges, one Waugh Model 8 sharpener, one Sullivan sharpener, two die racks, a tempering tank, an oil tempering tank, a steel truck, a rack for sharpened steel, a grinder, a coal-fired forge, and an anvil. Fuel oil is kept in a storage tank outside the mine, whence it is piped and pumped to the shops, concentrate dryer, and boiler house. Compressed air is used for atomizing. The coal forge and anvil are used by the drill sharpener for straigthtening steel and sharpening picks. The Sullivan sharpener is in the northwest corner, and the Waugh in the northeast corner, leaving the two oil-fired forges together between the sharpeners.

The car used to transport steel between the mine and shop is 3 ft. wide, 8 ft. long, and divided lengthwise into three compartments, each 1 ft. wide. This arrangement permits jackhammer, stoper, and drifter steel to be separated, and facilitates handling and sorting in the mine. The mine crew delivers the steel car to the top of the mill service incline. Shop men let it down the incline and make two deliveries each day from mine to shop and shop to mine. They also handle other equipment or supplies.

Steel delivered to the shop one day is cleaned, straightened, and laid on the shop-steel truck ready for sharpening next day. The steel truck is set between the two forges. A helper heats steel and hands it to the operator. After sharpening, it is laid on the rack directly behind him. At present, only the Sullivan sharpener is in use. The helper takes the sharpened steel from the rack and brings it to the proper temperature in the oil forge, and then hands it to the operator at the tempering tank. The shop is small for two machines, but it is not crowded. An operator and helper can handle between 300 and 400 pieces in an eight-hour shift.



Sketch showing method of stelliting bits



Special three-compartment car for transporting drill steel

Jackhammer steel is $\frac{7}{8}$-in. hexagon hollow, cut in following lengths: 3 and $3\frac{1}{2}$ ft. for $2\frac{1}{8}$-in. gage bits; 5 and $5\frac{1}{2}$ ft. for 2-in. gage bits; 7 and $7\frac{1}{2}$ ft. for $1\frac{7}{8}$-in. gage bits. Stoper steel is $\frac{7}{8}$-in. hexagon hollow, cut in the following lengths: $2\frac{1}{2}$ ft. for $2\frac{1}{8}$-in. gage bits; 4 ft. for 2-in. gage bits; $5\frac{1}{2}$ ft. for $1\frac{7}{8}$-in. gage bits; 7 ft. for $1\frac{3}{4}$-in. gage bits. Drifter steel is $1\frac{1}{8}$-in. round hollow, cut in the following lengths: 3 and $3\frac{1}{2}$ ft. for $2\frac{1}{4}$-in. gage bits; 5 and $5\frac{1}{2}$ ft. for $2\frac{1}{8}$-in. gage bits; 7 and $7\frac{1}{2}$ ft. for 2-in. gage bits.

A standard cross bit is used with all types of drills. At present only 1,500 bits per month are sharpened, which is much below the capacity of the shop, and the sharpener is operated only three times a week. Broken drills are less than 1 per cent of the number sharpened. All heats are determined by eye. Forging temperature, determined by color, is cherry red, and tempering heat, dull cherry red. Drill shanks are oil-tempered at dull cherry red. A good temper can be recognized by a slight scaling from the shank end, back about $\frac{3}{4}$ in. Shanks are dressed on the grinder before tempering.

No records of feet drilled per bit have been kept for the entire mine. A well-tempered bit will drill 8 to 10 in. in quartzite, 12 in. in granite, and 3 to 6 ft., or even more, in limestone. The ore is a massive sulphide deposit, consisting largely of pyrite and blende. Some of it is soft, causing little wear on a bit. Other sections of the deposit are harder and abrasive, quickly wearing away the gage of the bits.

In addition to the drills used in regular mining operations, the deep-hole prospecting equipment presents some interesting problems. Holes are drilled to considerable depth, and changing a bit on the end of a long string of rods involves time and expense. Rods used are $1\frac{1}{4}$-in. round hollow drill steel, and bits are $3\frac{1}{2}$ to $2\frac{1}{4}$ in., in $\frac{1}{8}$-in. changes. Since the stelliting process has been perfected, the number of changes has been reduced. Formerly the rods were threaded and used as $1\frac{1}{4}$ in., and some of these are still in use. A better practice has proved to be to upset the ends of the rods to $1\frac{1}{2}$ in., and cut the threads on the upset portions, using a $1\frac{1}{2}$-in. sleeve. The sleeves are purchased, but the rods are threaded in the lathe in the underground shops. After threading, the rods are tempered in oil at a low heat, to obtain hardness and toughness.

The rods are cut in 3- and 6-ft. lengths, the short rods being used for making changes. Space in the mine is limited, but, where practicable, 12-ft. rather than 6-ft. changes would seem advisable. The Waugh sharpener has been rigged with die and hammer for upsetting the ends, and the operation is now as simple as shanking a piece of steel. Repairs to rods with upset ends are rare. On the smaller rods the ends chip, threads on rods and sleeves wear, and rods break at the base of the threads. When rods with chipped ends come to the shop, they are first annealed and cooled in lime, then squared off in the lathe, and threads trued up and run down further. The only remedy for worn threads and broken rods is to cut new threads and furnish new sleeves. After cutting threads it is, of course, necessary to retemper the rod in oil. After a string of tools has operated for some time, annealing of the ends and retempering have been found advisable.

Standard cross bits are made and sharpened in the shop. After shaping in the sharpener, the cutting edge is ground off until it is $\frac{1}{8}$ in. wide, and the gage is ground down $\frac{1}{8}$ in. on each wing, as illustrated in an accompanying sketch. Stellite is then applied to fill in the portion ground off, and also to cover the bevel face $\frac{1}{16}$ in. thick. Properly applied, this should give a smooth even surface; any high spots that may occur

are removed with the emery wheel. The gage particularly must be left true. The stellited bit is then heated and tempered in oil. Threads on bits are 1¼ in., and an adapter having 1¼-in. threads on one end and 1½ in. on the other is used. Lately a few bits are being tried with 1½-in. thread, making use of the adapter unnecessary.

One foreman, one blacksmith, one drill sharpener, one helper, seven repairmen, and two repair helpers constitute the shop crew. These, with one man in the mine, handle the shop, and the mine and mill repairs of the plant.

The electrical shop is located on the floor above the blacksmith and sharpening shop. An overhead I-beam is used for hoisting motors or other equipment from the service track to the shop. The shop equipment includes a work bench, drawers for small tools, steel cabinets for supplies, instruments, and necessary test panels. All ordinary motor and electrical equipment repairs are made in the shop. Motors are sent outside for rewinding. The crew consists of one head electrician, three electricians, and two helpers—a small force, when one considers the work done.

The underground situation of the shops has proved notably satisfactory. In addition to other lights, Cooper-Hewitts are used over the lathes and in the blacksmith and drill sharpening shops. Two small steam radiators were provided for workmen near openings in winter; otherwise air temperature in the shops is uniform. Heating cost is low, lighting is excellent, ventilation good, and maintenance of the shops costs practically nothing.

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Trench Digger Mines Gypsite in Arizona

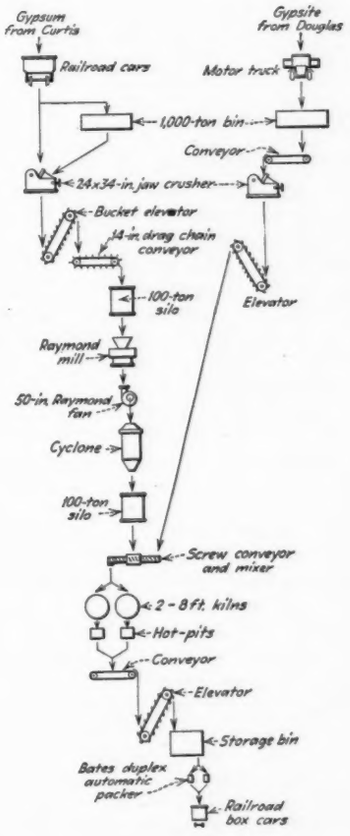
J. B. Huttl
Assistant Editor

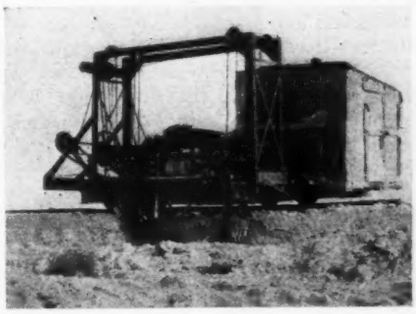
DURING the last 30 years the Arizona Gypsum Plaster Company has operated steadily in Cochise County, Ariz., producing wall plaster, locally known as Douglas plaster; gypsum building blocks; agricultural gypsum; and partition tiles. Raw gypsum is obtained from the company's deposit at Curtis, Ariz., about 80 miles northwest of Douglas, Ariz., where the company controls a tract of more than 1,500 acres. Gypsite for blending is mined from a flat deposit, occurring in a wide wash, 6 miles east of Douglas. The processing plant at Douglas is connected by a short spur with the main line of the Southern Pacific Railroad.

The mine at Curtis consists of several open cuts that vary in depth according to the contour of the surface. Devoid of overburden, the exploitation of the deposit is simple. The gypsum is blasted from the ledges with 15 per cent dynamite, manufactured specially for this work by the Apache Powder Company, and stored in wooden loading bins. From here it is conveyed by trucks to Curtis for shipment to the plant at Douglas. Large boulders and slabs are reduced by hand to 6-in. The broken material is of a grayish-white color, hard, and spotted with gypsum crystals. It analyzes: calcium sulphate, 78.74 per cent; water, 19.70; ferric oxide, 1.1; and silica, 0.46 per cent.

The gypsite deposit near Douglas, averaging 4 ft. deep, is mined by a gasoline-driven Buckeye trench digger, mounted on a flat four-wheel truck operating on a movable track laid along the length of the deposit. This digger discharges the excavated material into the waiting truck. The time to excavate and load 10 tons is about 15 min. An additional 30 min. is required to truck the loaded gypsite to the 1,000-ton storage bin at the mill.

From the standard railroad cars the gypsum is stored in the 1,000-ton bin,





made of reinforced gypsum, or fed direct to an Ehrsam 24x34-in. jaw crusher. The crushed 2-in. material is raised by a bucket elevator and discharged onto a 14-in. chain drag, to be conveyed to a steel flat-bottomed silo with a capacity of 100 tons. From here it is evenly fed to a five-roll Raymond mill, from which the fines are drawn off by a 50-in. Raymond fan, then forced through a cyclone and discharged into a superimposed 100-ton silo. The pulverized product falls by gravity upon a screw conveyor, equipped with a home-made proportioning apparatus for gypsite, which conveys the mixed gypsum and gypsite to the calcining department. Two oil-fired Ehrsam kilns, each 8 ft. in diameter, are available. Each kiln contains an agitating mechanism, to insure uniform curing of the charge, and a series of dampers to regulate kiln temperature. Temperature-control equipment consists of two automatic recording pyrometers. Moisture of crystallization contained in the raw material is reduced from 19 to about 4 per cent.

Cured gypsum from the kilns drops into the adjoining hot-pits, whence it is conveyed by a bucket elevator and a belt conveyor to the main storage bin, equipped with a Bates duplex automatic packer. The finished material, of which 85 per cent will pass a 200-mesh sieve, is packed into five-ply paper bags for shipment to Arizona and New Mexico.

Gypsite from the 1,000-ton wooden storage bin is belt-conveyed to a small crusher, then raised by a bucket elevator and discharged into the proportioning apparatus attached to the screw conveyor which feeds the pulverized gypsum to the curing kilns. All plant machinery is electric driven. Building blocks and partitioning tiles are made by hand, with wooden forms, in a building adjacent to the gypsite storage bin S. G. Dowell is general manager of the company.

Science and Principles of Geophysics

Three Reviews by Dr. C. A. Heiland

HANDBUCH DER GEOPHYSIK. *By Prof. Dr. B. Gutenberg. Band III, Lieferung 1. Pp. 570; illus. Berlin: Gebrüder Borntraeger; 48 Reichsmarks.*

This volume is chiefly geological. The dynamics of the earth, chiefly the dymanics of its surface, are discussed. In the first section the author deals with forces produced by temperature changes, meridional and equatorial land drift, pole displacements, and the like. The next section represents an exhaustive treatment of all phases of the volcanism of the deep and surface rocks. Von Wolff discusses first the constitution of the earth's interior and the physical and chemical properties of the various types of magmas. He then treats of the formation of the plutonic rocks, and their geologic appearance. A chapter on submarine eruptions is followed by a detailed description of surface volcanic phenomena, their relation to plutonic processes, the rock products of volcanic activity, and of numerous examples of fissure and crater eruptions. In the last chapters, the gases liberated by explosions, and their products, the postvolcanic phenomena, and the geographic distribution of the now active volcanoes, are discussed.

By far the major portion of the next section—on movements of the earth's crust—deals with the epeirogenic movements, and discusses such topics as the relation of rock texture to mechanical stresses, the types of folding, structure of mountain regions and the amount of horizontal compression in the same, the nature of geosynclinal regions, the migration of folding and the influence of the subcrustal rocks and of the rigid frames on the arrangement of folds. Two additional chapters contain details on the relation of geologic structure to commercial deposits, and experimental investigations concerning structural processes. The chapter on epeirogenic movements discusses topics such as the geologic indications of such movements, and their causes.

A section on geotectonic hypotheses presents an exhaustive and clearly written summary of the numerous theories that have been advanced since the early beginning of geological science, to account for the existence of our planet, for its morphologic and geologic constitution, and for the forces acting on its surface. The author begins with a discussion of the geophysical foundations upon which such hypotheses are built, such as pole shifts, continental drift forces, epeirogenic and orogenic movements, isostasy, temperature and pressure in the interior, and elastic phenomena. The description of the geotectonic hypotheses which then follows is divided into cosmogonic hypotheses, hypotheses that are based upon chemical phenomena, sedimentation and erosion, contraction and expansion, tides, pole shifts, and continental drift.

The fifth section of the book represents the first part of a description of the effects of ice, water, and wind on the surface of our planet, and deals with the mechanical effects on the morphology produced by glaciers and polar ice caps.

HANDBUCH DER GEOPHYSIK. *By Prof. Dr. B. Gutenberg. Band IV., Lieferung 1. Pp. 298; illus. Theorie der Erdbebenwellen; Beobachtungen; Bodenunruhe. Berlin: Gebrüder Borntraeger; 22 Reichsmarks.*

This volume deals with the theory of the various types of elastic waves, presents a discussion of the observations, and gives data on ground unrest as observed on seismic stations. The author makes an exhaustive presentation of this subject. The first part, on the theory of seismic waves, fills a long-felt want in seismic literature for a comprehensive treatment of this involved subject. The type is clear, the representation of graphs, figures, and mathematical formulae is excellent. In view of the outstanding nature of the content and of the appearance of the book, the price is exceptionally low.

In the introductory chapters, the author deals with the fundamental theory of elasticity in a homogeneous elastic medium, and the propagation of waves as dependent upon the elastic constants. The general relations are then expanded to the case of the viscoelastic medium, and such topics as absorption, dispersion, and group velocity are treated. Then follows a chapter on the paths of the elastic waves in the earth's interior, and the various methods for the determination of wave velocities and discontinuities are discussed. The effects of discontinuities on the various types of waves are analyzed, and methods for considering the depth of the focus are given. The next chapter deals with the theory of surface waves, their dispersion and group velocities, oscillations of layers, amplitude relations, and travel-time curves.

The second section of the book presents the results of observations of various types of earthquake waves, with reference to the first and second forerunners, their velocities in various parts of the earth's interior, and their amplitudes as dependent upon path and distance. Data are given on the longitudinal and transverse waves for near quakes. A chapter deals with periods, amplitudes and travel time curves of surface waves. The third and last section of the book presents data on periods and amplitudes of ground unrest, and a discussion of causes.

HANDBUCH DER GEOPHYSIK. *By Prof. Dr. B. Gutenberg. Band IV., Lieferung 2. Pp. 386; illus. Dr. H. P. Berlage, Jr.: Seismometer, Auswertung der Diagramme; Prof. Dr. A. Sieberg: Geologie der Erdbeben. Berlin: Gebrüder Borntraeger; 30 Reichsmarks.*

After reviewing the most important types of seisometers, the author discusses first their common features of construction, with chief reference to the construction of horizontal and vertical seismographs, and methods for recording and damping. He presents his own general theory of seismographs in an excellent manner, and derives the fundamental relations which permit the precise determination of the ground movement from the seismograph records. Diagrams are given for the dymamic magnification of seismographs for various ground frequencies, and also for the phase shift. The theory is then applied to the special case of the horizontal seismograph, and the mechanical, optical, and galvanometric method of recording are each analyzed in detail.

The next section presents a complete and well-illustrated description of all types of seismographs, together with instructions as to the manner in which they are to be set up and their constants determined. The last section of the first part of the book deals with the interpretation of seismograms, and methods for the determination of the true ground movements from the records, as well as the procedures followed to determine the azimuth and coordinates of the epicenter, and also the depth of the focus from the records.

The second part of the book describes the geology of earthquakes—their origin, classification according to cause, propagation of the energy, and the speeds of waves in formations of various geologic characters. This is followed by a discussion of the methods for the determination of earthquake intensities, and their relation to the geologic structure. The damage of earthquakes to buildings and their prevention, and the effects of earthquakes on watercourses, are discussed, and then a detailed description of structural and volcanic quakes is given. The final chapters deal with sound and light phenomena observed in conjunction with earthquakes. Detailed instructions as to how to record macroseismic observations are appended.

Guide to Flotation Practice

A Review by Dr. Rudolf Gahl

FLOTATION. *By E. W. Mayer and H. Schranz. Leipzig, Germany: S. Hirzel. Pp. 593; illus. Price 33 marks; bound 35 marks.*

This volume forms part of the series "Chemie und Techniker Gegenwart," edited by Professor Walter Roth. To quote from the preface: "Need of a fairly complete description of flotation and associated processes rose in the same degree in which flotation developed industrially and scientifically. It seemed particularly desirable that a book should be written which would give special attention to the smaller although in many cases more difficult applications of the process in Europe, without neglecting the larger American installations." The volume presented, to fill the existing gap, may be said to do this well. It goes beyond the treatment of subjects the discussion of which might logically be expected, such as history, theory, and machinery of the flotation process. As an example may be cited a discussion of detailed flowsheets, with particular reference to grinding and classification preliminary to flotation, as well as of flowsheets used in connection with flotation machines proper; an outline of the various flows employed in the treatment of middling, of rougher concentrate, and other products. Use of tables as auxiliaries to flotation machines is also treated in similar manner.

Of particular interest is the stand taken by the authors in dealing with the theory of flotation. Their views in regard to the theory of contact angles and their relation to flotation appear to be assailable. The authors are under the impression that this theory was abandoned because it failed to explain many phenomena in modern practice. But has it been abandoned? With equal logic the claim may be made that the theory is as firmly grounded as ever. In fact, the conclusions of what the authors call the capillary theory might as well be stated in terms of contact angles. One fact is evident: actual experimentation with contact angles has proved extremely difficult and uncertain; testing of the theory by actual measurements has been unsuccessful and may perhaps be said to have been discarded.

The authors adopt the theoretical terms that for several years have played a leading rôle in American flotation literature in speaking about polarity and non-polarity of reagents and in attributing the action of collectors to combinations of polar and non-polar groups in the molecule, according to views developed in the United States. The question remains, however, if, in adopting these terms, they have taken over anything of actual value to the flotation operator. Does the flotation engineer derive any help from the axiom that the surfaces of the floatable minerals attract certain polar groups, and repulse non-polar ones. Would not adherence to the older nomenclature be much better, to speak about hydrophil groups, perhaps enlarging this nomenclature by adding the terms metalophil and aërophil? Without unnecessary theorizing, the statement may be made that a collector presents a combination of metalophil and aërophil groups, whereas a frother is a combination of aerophil and hydrophil groups.

Naturally, the review on flotation history emphasizes the part played by German scientists and technicians in the development of the process. We learn for instance (what does not seem to be known in the United States) that as early as 1877 the Bessel brothers, in Dresden, took out a patent (German patent No. 42) for flotation of graphite, in which the lifting of the graphite particles was accomplished by means of air bubbles produced by boiling the pulp after addition of small quantities of oil; in a later patent (German patent No. 39,369) they used air bubbles produced by the action of acid on carbonates or metals. The first patent by far antedates Carrie Everson's famous patent.

The book under review may be classed as a highly meritorious work, which undoubtedly will find many interested readers even in the English-speaking countries where no scarcity of flotation literature exists. American books on flotation have almost without exception been written by college professors and others inclined to stress the theoretical side of the subject. Flotation operators and designers have so far been slow in presenting their experience in book form. For this reason, perhaps, the existing books on flotation have a greater appeal to the student of the process than to the man in the field. Mayer and Schranz's "Flotation" may be preferred by the man actually floating the ore or building plants for doing so.

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INDUSTRIAL ELECTROCHEMISTRY. *By C. L. Mantell. Pp. 528; illus. New York: McGraw-Hill Book Company; $5. London: McGraw-Hill Book Company, Ltd.*

LACK of a comprehensive American textbook covering the entire field of industrial electrochemistry has prompted the author to prepare this addition to the Chemical Engineering Series. The subject is too extensive to permit detailed treatment of the multiple branches of the industry within the confines of a single volume. As a textbook and for general reference work, however, the presentation should be of great value.

An interesting feature of the work is the large number of tables, giving comprehensive data, summaries, and comparisons of the processes described. These tables assist the reader to grasp quickly the essential features of the various operations treated. Those of the tables that deal with electrolysis in aqueous solution were presented in a recent issue of this publication.

The book is divided into six parts. Initial chapters deal with theoretical electrochemistry. Part II discusses corrosion, oxidation and reduction, colloid chemistry, electrolytic cells, and accumulators. Electrolytic processes are treated in Part III. Among the subjects dealt with in this section are electroplating; electrolytic refining; electrolysis with insoluble anodes; alkali-halides electrolysis; and processes treating fused salts. Electric furnaces, electrochemistry of gasses, materials of construction comprise the remaining chapters.

Obsolescence is often unavoidable in a rapidly moving age. In the chapter on electrolytic zinc, however, the change in conditions resulting from advances in selective flotation has not been taken into consideration, although these advances came several years ago. The chapter on nickel does not bring out clearly how the process as it appears today is a logical development of the Hybinette process, with addition, in the preliminary stages, of certain features of the old Orford process. As presented, the reader is given the impression that two distinctly separate processes exist.

These are, however, minor details, which do not detract from the merit of the volume. The author is to be congratulated on the able manner in which he has completed his task. Numerous illustrations and a comprehensive index add to the value of the book.

B. H. STROM.

WHO'S WHO IN ENGINEERING. 1931. *By W. S. Downs and M. M. Lewis. Pp. 1536. New York: Lewis Historical Publishing Company, Inc.; $10.*

This is the third edition of an indispensable work of reference. Its value is enhanced by the fact that qualification for admission was determined according to the recommendations of an advisory committee appointed by American Engineering Council. It therefore includes (1) Engineers of outstanding and acknowledged professional eminence; (2) engineers of at least ten years' active practice, at least five years of which have been in charge of important engineering work; (3) teachers of engineering subjects in colleges.

Among Contributors To This Issue

Livingston Wernecke (University of Washington) acquired a wide experience in the Northwest before acting as construction engineer for the Alaska Gastineau company in 1914. Followed geological and mine work for the Alaska Treadwell company (1915-20), leading to his present position—manager and general superintendent of Treadwell Yukon, Ltd., held since 1921, with headquarters at Wernecke, Yukon Territory.

Walter A. RuKeyser (Princeton and Columbia School of Mines) has held consulting and executive positions in Canada, Mexico, and the United States, specializing in asbestos mining and processing. In recent years he served as consultant to the U.S.S.R. state trust "Uralasbest," in charge of asbestos developments in Russia.

Morris J. Elsing (Columbia School of Mines) is a consulting engineer, with headquarters at Tucson, Ariz. He was formerly geologist to Copper Queen Consolidated, and later in charge of the geological department and of the mining operations of Cananea Consolidated, prior to acting in a consulting capacity to that corporation. He recently contributed to this magazine a series of articles on mining methods.

T. H. Garnett, a graduate of the Colorado School of Mines, is engineer in charge of service and maintenance at the Gilman property of the Empire Zinc Company. He was formerly general superintendent of the Mineral Point Zinc Company, another subsidiary of The New Jersey Zinc Company. Mr. Garnett supervised the construction of the underground mill at Gilman, which was described in a recent issue.

Lucien Eaton (Harvard) was for fifteen years superintendent of the Cleveland-Cliffs properties in the Lake Superior iron region. He made notable contributions to the technique of mining in that region, and in recent years has occupied the position of consulting mining engineer on the staff of Rhodesian Selection Trust, with headquarters in London, from which center he recently returned to the United States.

J. B. Huttl, assistant editor, was educated at the M.A.N. Technical School and the Stadtische Bauschule, Nürnberg, Germany. He has had an operating experience with mining companies in the Ruhr district of Germany, in Spain, Mexico, and Central America, specializing in mechanical engineering and plant maintenance.

Ernest Raymond Lilley is professor of economic geology at New York University, his alma mater. He is the author of several books and numerous articles on the economics and geology of the petroleum industry. In 1930, having been awarded a fellowship by the John Simon Guggenheim Memorial Foundation, he went abroad, on sabbatical leave from the university, for a year's study of mineral resource problems in Europe, whence he returned last July.

M. S. Hansen was graduated from the University of Utah in 1927. Subsequently he was appointed an instructor in metallurgical engineering there and held other educational posts. He is now research engineer in ore dressing for the Reilly Chemical Company, Indianapolis, Ind.

A. H. Hubbell, associate editor (College of the City of New York and Columbia School of Mines), obtained operating experience with Tennessee Copper, Portland Gold Mining, Utah Copper, South Utah Mines & Smelters, and other companies.

Rudolf Gahl (Göttingen and Berlin universities, Germany), consulting metallurgist, well known as an authority on the treatment of copper ores, has been associated with Phelps Dodge Corporation, Inspiration Copper, and Cerro de Pasco companies in research and operating capacities.

W. R. Meyers (Lehigh University) joined the staff of the Cleveland-Cliffs Iron Company in 1907. He now holds the position of superintendent of the Tilden mine—a siliceous open pit—at Ishpeming, Mich., and the Spies-Virgil mine, at Iron River, Mich.

C. A. Heiland (Hamburg and Heidelberg universities, Germany) has practiced as a geophysicist in Europe and in the United States. Since 1926 he has occupied the post of professor and head of the department of geophysics at the Colorado School of Mines.

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Huntington Adams is at Majorca.

Guy C. Riddell was recently in Vienna.

Frank D. Pagliuchi returned to Venezuela, after a short stay in New York, on Jan. 2.

A. W. Beam, of Benguet Consolidated, P. I., recently passed through San Francisco en route to New York.

W. J. Freeman recently retired as superintendent of International Nickel Company at Port Colborne, Ont., Canada.

George A. Kervin has returned to California from Namaqualand, where he was in charge of development of South Africa Copper, Ltd.

Sewell L. Avery, president of U. S. Gypsum, has been elected chairman of the board of Montgomery Ward & Company.

Robert M. Catlin, who recently resigned as manager of New Jersey Zinc's operations at Franklin, N. J., has retired and is living at Piedmont, Calif.

Ira B. Joralemon writes from San Francisco to correct an error in his article on Pioneer Gold, which appeared in a recent issue. The second and third sentences in the third paragraph on the third page should have read: "The ore expected per 100 ft. additional depth, based on the 800 level, is 54,000 tons. It should be nearly 65,000 tons per 100 ft. additional depth below the 9th level."

Walter Curran Mendenhall recently became director of the U. S. Geological Survey, on confirmation of his appointment by the Senate. Previous directors of the Survey, established in 1879, comprise Clarence King, John W. Powell, Charles D. Walcott, and George Otis Smith. For more than a year Mr. Mendenhall has been serving as acting director. This was due to the fact that Doctor Smith, when he was appointed chairman of the Federal Power Commission, expected to return to the directorship of the Survey as soon as a permanent chairman of the Commission had been appointed. But he was assailed by a group in the Senate, out of which has grown a lawsuit that bids fair to result in a decision as to the respective rights of the executive and legislative branches of government. With that important question before the courts, Doctor Smith has not felt at liberty to retire from his present position, and is understood to have abandoned thought of returning to his old post. His successor, Mr. Mendenhall, is thoroughly familiar with the work of the Geological Survey, having been in its service for more than 30 years. He was in charge of ground-water investigations from 1907 to 1910, and was appointed chief geologist in 1922. Mr. Mendenhall was born at Marlboro, Ohio, on Feb. 20, 1871. He was graduated from Harvard in 1897 and from the University of Heidelberg in 1900.

W. C. MENDENHALL

R. L. Peek, consulting metallurgist to International Nickel, has moved his headquarters from Toronto to Copper Cliff, Ont.

F. W. Griffin is en route from San Francisco to New Guinea, where he will inspect construction operations of the Bululo Gold Dredging Company.

James E. Fisher, formerly secretary-treasurer of Calumet & Arizona Mining, at Calumet, Mich., has been made assistant secretary of Phelps Dodge Corporation.

Sir Newton Moore, of London, a director of the Great Boulder Gold Mining Company, is visiting the company's mine at Kalgoorlie, Western Australia, in company with **A. H. Collier.**

J. W. Rawlins is now technical assistant to the general manager of International Nickel, at Copper Cliff, Ont. Mr. Rawlins was formerly stationed at the company's Port Colborne refinery.

W. G. Andrews, managing director, Bells Asbestos Australia Agency, is in Australia in connection with the erection of a factory to manufacture asbestos products from Australian asbestos.

T. L. Willan has severed his connection with Anglo-Oriental Mining Corporation and will engage in examination and consulting work on his return from London to Malaya in the near future.

J. C. Nicholls, formerly general manager of International Nickel at Copper Cliff, Ont., has assumed his duties as assistant to the president of the company, with headquarters at Toronto, Ont.

A. H. Richards, superintendent of the A. S. & R. Smelter at Tacoma, Wash., recently delivered an address before the Columbia section of the A.I.M.E., at Spokane, Wash., on "The Future of Copper."

J. J. Andrews, mining inspector attached to the Tasmanian Department of Mines and stationed at Queenstown, has been appointed general manager, North Mount Farrell Company, operating in Western Tasmania.

W. Kingston, of the Geological Survey of Victoria, Australia, has been appointed to take charge of the government's prospecting scheme by which unemployed men are granted assistance and educated in prospecting.

K. L. Bockman, whose article on "Direction Lines for Drift Curves" appeared in a recent issue, calls attention to an error. In the middle column of the first page, penultimate line, is an equation: $y = x_2 \div 2r$. This should read: $y = x^2 \div 2r$.

A. G. McGregor is the author of a book "World Depression to World Prosperity," recently published by P. S. King & Son., Ltd., London, in which he maintains that "the purchasing power of our bank notes should remain constant, and should be regulated to correspond exactly in exchange value with the average value of all the useful commodities." At the conclusion of an interesting study of financial and economic conditions, he concludes by averring that "By extricating itself from the toils of gold, and establishing a currency of constant purchasing power at equitable price levels within its own confines, each nation would enable natural economic forces to function in automatically casting off depression. . . . "

Camp on the concession of the Compagnie Equatorial de Mines, in the diamond-bearing region of Brea, French Equatorial Africa. Pierre Delaitre (left) is general manager. John L. Middleton (right), chief geologist

P. C. Merrill, who recently returned from the Urals after the completion of a two-year contract with the U.S.S.R., is in New York.

Byron M. Johnson—until recently metallurgical engineer on the staff of the Kazakstan Copper Combine of the U.S.S.R., northeastern Turkestan—has left Russia, and is now in Rome, Italy.

N. J. Everard, of Haileybury, Ont., is now manager of the Moss Gold Mines at Kashabowie, Ont. Mr. Everard was formerly manager of the Crown Reserve at Larder Lake; and, previously, of the Vipond Mine at Porcupine.

F. C. Lincoln, professor of mining at the South Dakota School of Mines, contributes the introductory article of the November issue of *The Black Hills Engineer,* devoted to the history and practice of placer mining.

T. H. Wickenden has been promoted to the position of assistant manager of development and research, International Nickel Corporation; and **H. J. French** has been transferred from the laboratory at Bayonne, N. J., to the development and research department in New York.

A. G. Langley, consulting engineer of Vancouver, B. C., was recently elected vice-president of the Association of Professional Engineers of British Columbia, and was nominated a vice-president (representing British Columbia) of the Canadian Institute of Mining & Metallurgy for 1932.

H. S. Denny, a metallurgist and mining engineer of international experience and reputation, is the author of an exceptionally informative and lucid bulletin (No. 80, Department of Mines, Province of Ontario), on "Money and the World Crisis," in which strong arguments are advanced in favor of bimetallism.

Ernest Williams has resigned as general manager of Great Boulder Perseverance, Kalgoorlie, Western Australia, and is succeeded in that position by **C. E. Blackett,** formerly superintendent of erection of the new bromo-cyanide plant, to be completed in nine months, and so arranged that it can be converted to a flotation unit if necessary.

Michael Curley, general manager of New Cornelia mines, Phelps Dodge Corporation, was recently elected governor of the Arizona Mining Congress; **William Koerner,** general manager, Magma Copper, Superior, vice-governor; and **W. Val DeCamp,** general manager, United Verde Copper, Jerome, second vice-governor.

John C. R. C. Pape, for the past six years with Chile Exploration, at Chuquicamata, Chile, was recently in New York, en route to Mexico, where he will join some of the sourdough friends with whom he was associated in Alaska, for a prospecting trip. Correspondence, which will be forwarded, may be addressed to P. O. Box 766, Longview, Tex.

C. F. Williams, mining supervisor, U. S. Geological Survey, Miami, Okla., who uses an airplane in connection with his official duties, writes that the cost per mile, excluding storage, of air transport in his work is less than the average cost for the government-owned cars used by the Geological Survey, and that the total cost, including storage and depreciation, compares favorably with the automobile records.

Dr. G. Horn, geologist for the Norwegian Spitzbergen Office, is on a three months' lecture tour in the United States. For the last seven years Doctor Horn has been making geologic surveys of the Norwegian possessions in the Arctic Ocean. During one of these expeditions, in the summer of 1930, his party received wide publicity when, during a visit to the White Island, they discovered the remains of the ill-fated Andrée balloon expedition. Doctor Horn is a graduate of the Norwegian Institute of Technology. He has also studied at the Royal School of Mines in London and the Institute of Technology in Berlin. Prior to his connection with the Spitzbergen Office he spent three years in Trinidad and Venezuela.

J. Brian Eby, at a recent meeting of the American Association for the Advancement of Science, at New Orleans, stated that the discovery, by torsion balance and seismograph, of more than sixty salt domes on the Gulf Coast of Texas and Louisiana during the last eight years was a remarkable achievement. "The potential valuation of the coastal area of this region has been more than doubled," he averred. "Oil and sulphur, the only valuable minerals now exploited on the coast, are being produced from these newest discoveries in increasing quantities. The torsion balance, although invented more than thirty years ago by an Austrian, Roald von Eötvös, was not introduced into the Gulf Coast until 1922."

R. L. Suhl has been appointed manager of the nickel sales department of International Nickel. Mr. Suhl joined the original company in 1903, became superintendent of the electrolytic refining plant at Bayonne, N. J., in 1908, and was superintendent of the oxide division, in charge of roasting and leaching, during the War. In 1920 he joined the headquarters staff in New York as development engineer. In 1928 he became assistant sales manager of the nickel department. **Ransom Cooper, Jr., Walter C. Kerrigan** and **Charles McKnight** have been appointed assistants to Mr. Suhl.

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Great Bear Lake

In a recent issue *The Northern Miner* discusses an alleged interview published in a New York newspaper that, in the absence of denial or disavowal by the persons mentioned therein, deserved adverse criticism because of extravagant statement and absurd claims. *Engineering and Mining Journal* commented on this same report in an editorial, without in any way casting doubt on the discoveries in the region in question—Great Bear Lake.

To discourage publicity that is harmful to expansion of the mining industry and detrimental to the profession of mining engineering is one of the obligations of journalism that this publication has observed since its foundation more than sixty years ago. Our Canadian contemporary has courteously sent us a clipping from *The Northern Miner* of Nov. 5, asking us to note its reactions to this type of publicity. *Engineering and Mining Journal* is glad to republish this in supplement to its own criticism:

"New York is hearing about the pitchblende-silver finds of Great Bear Lake. On the assumption that the New York news palate is insensitive to anything less highly seasoned than caviare Great Bear is being served up in style, and plentifully garnished with trimmings of Bunyanesque quality.

"The New York *World-Telegram* in a recent issue reports an interview with E. E. Campbell, Toronto mining engineer, who is on a visit to the city of ballyhoo and who relates some facts relative to Great Bear. The New York paper puts it this way:

"'E. E. Campbell, veteran mining engineer and rugged northerner, who now lives in Toronto, leaned forward in an easy chair in the apartment of his friend, H. P. Davis, at the Brevoort Hotel, and expatiated on the wonders of the Great Bear region, where now is unfolding the latest thrilling chapter in the world's romance of mineralogy. "I haven't seen it yet with my own eyes," admitted Mr. Campbell, "but two of my men have been up there and we have some claims staked out. The men went in by airplanes. One of them has made camp there for the winter." Next spring, Mr. Campbell predicted, will witness a "rush" to the Great Bear Lake country that will outdo the rush to California in 1849 or the world-wide migration to Cobalt, Ontario, in 1905. The new mineral find, news of which has already spread round the world and to the call of which grizzled prospectors from Alaska to the Transvaal are harkening, was made in the barren wastes, exactly on the Arctic Circle, Yellowknife Preserve, District of Mackenzie, in the Canadian Northwest, Mr. Campbell said. Mr. Campbell's one representative with the score or more others who stayed behind when the last airplanes winged away to the south will live in igloos through the winter, catching a brief glimpse of the sun each midnight, subsisting on meager tinned rations and battling the intense cold with fires made from logs conveyed to the camp by aerial express last summer at a cost of $25 a log.'

"The price of logs must have dropped lately," remarks the writer in *The Northern Miner*, as the last market reports from the *Arctic Gazette* quoted $50 a log. And logs up there are not big at that. Burning a log a day during the 200 odd days of winter the lone prospector in his igloo would use up $5,000 worth of fuel in a season. It is quite evidently no piker's camp. There is quite a trick, too, in burning logs in an igloo, because igloos are temperamental things, melting on slight provocation and nothing will put out a fire quicker than a collapsed snow house. To those unfamiliar with Arctic conditions of prospecting it may seem far-fetched to talk about delivering logs by aerial express—at $25 a log. Yet in this case fiction is considerably weaker in the knees than fact. The N.A.M.E. crew actually flew logs in to Dismal Lakes for the purpose of shoring up trenches.

"Like Mr. Campbell we have not seen the thing with our own eyes but we are credibly informed that the technique of delivering timber by air to the scattered igloos of the lone, tin-can eating prospectors has been highly developed. Most of the pilots operating on the Arctic Circle patrol are ex-army fliers, with bombing training. They have rigged their ships with bomb-droppers and, as they circle over the said lone prospector's igloo they release the trigger and drop the log. Deliveries are made at five a.m. and six p.m. daily except Sundays. Log tickets are collected once a month. The igloo dweller hangs out a sign 'Two Logs Wanted' or 'No Logs Today' according to the condition of his supply of fuel.

"We are particularly pleased to meet in this New York story our old friend, the 'grizzled' prospector. The yarn would not sound right without him. There is something about the business of prospecting which grizzles its devotees without fail. Actually there are times when you can hear him grizzling, as for instance on a soggy portage in fly season when his tump line breaks. Sometimes he grizzles in Spanish, sometimes in a language which sounds like Siwash but mainly in good homely English with occasional biblical references and a glossary of mule driving terms. Once the writer saw a prospector who wasn't grizzled but he was looked upon by his fraternity as a freak, somewhat in the same fashion as the natives of the South Sea islands regard an albino. He wasn't a good prospector either, being handicapped in that way.

"There was an organization in Toronto which a few years back took on a lot of prospectors and in considering the applicants the boss invariably asked the question: 'Are you grizzled?' And if the job seeker could not answer in the affirmative his case was not even considered. After all one must draw the line somewhere.

"Oh, well, they say that any kind of publicity is good publicity and if that is the sort of news that New York likes it may succeed in arousing some interest in Canada's newest mining field. However, it would appear to be advisable to bring your own logs with you when you head into Great Bear."

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PERSONALS constitute an attractive feature of a magazine of this type. Probably the most interesting ones record changes in employment. If a reader is promoted in his organization, or leaves one company to join another, he should keep his scattered friends informed of his movements. Of only negligible interest to the mining fraternity are notifications that short inspection trips and vacations have been taken. Engineers, particularly mining engineers, are normally peregrinating creatures. News items of journeys to little-known districts, or in connection with important missions, are most welcome, especially when accompanied by short personal comment of an informative or critical nature. When notifying us of change of address, the subscriber is urged to say a word or two about himself or one of his friends. At other times, send a note direct to the Editor, remembering that personal or pictorial photographs are occasionally used to lighten the text.

Agusto Genín—An Appreciation

On Dec. 3, 1931, in Mexico City, died one of Mexico's most versatile men—Agusto Genín—who, in addition to mining and other business activities, found time for scientific studies and literary pursuits. At the time of his death he was director of the San Rafael Mining Company, which he organized, and with which he had been connected for many years. He was born in Mexico, of French parents, during the Second Empire, was schooled in France, and spent his mature years in Mexico. Aside from his business, he found time to study Mexican archaeology, becoming an authority on the customs of the Indians before the Spanish Conquest. His monograph on the Aztec feasts, published by the Smithsonian Institute, is a classic. He wrote, with equal ease, Spanish, French, and English; and he published poems in all three languages, as well as one important historical novel in Spanish—"El Robinson Español."

FREDERICK MACCOY.

Berkeley, Calif.

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George Max Esterly, manager of the Llano de Oro Mines, Waldo, Ore., died Dec. 2, 1931.

Charles Minear, mine superintendent at Copperopolis, Calif., was found murdered in his automobile, near San Andreas, Calif., Dec. 21, 1931.

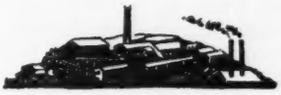
Francis H. Blake, mining engineer, died at Litchfield, Conn., on Dec. 30. His father, W. P. Blake, conducted the survey on which was based the purchase of Alaska by the United States.

Montague Cohen, director of Electrolytic Zinc, Amalgamated Zinc, and Minerals Separation, died recently in Melbourne. Mr. Cohen was 75 years old. He was associated with A. De Bavay in the application of flotation to zinc ores at Broken Hill.

Sidney Bradford, general manager for the Avery Salt Company, New Iberia, La., died there of heart failure on Dec. 29. Mr. Bradford was born in Switzerland, educated at Cambridge and Tulane universities. He had been general manager for the company since 1903.

Frederick Capel Brown died on Nov. 29, 1931, at Boise, Idaho. He began his career in the Black Hills, at the Homestake mine, subsequently gaining experience in Colorado, Idaho, and Mexico. Later he was sent by London interests to New Zealand, to operate properties there. He was the inventor of what is known as the Pachuca vat—an agitator for cyanide pulp, using compressed air in a manner that eventually permitted series treatment and continuous operation. The Komata tube-mill lining was another of his achievements. He resided in Boise for the past twenty years.

INDUSTRIAL PROGRESS

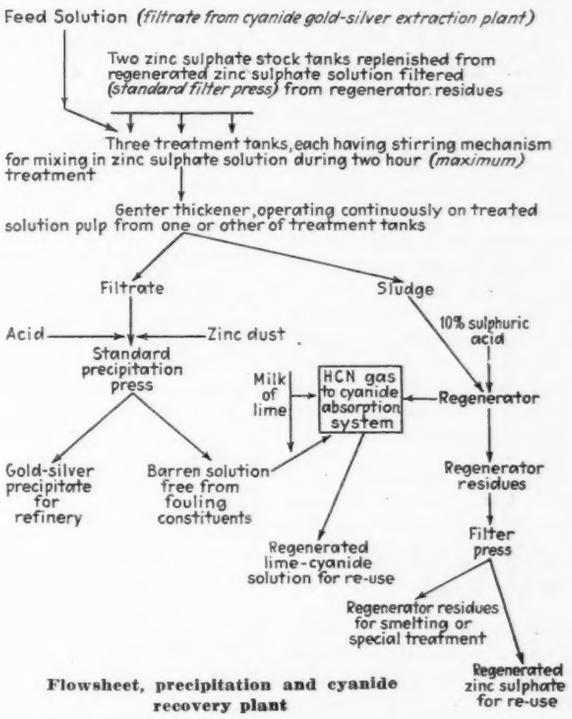


Precipitation and Cyanide Recovery Process Developed by General Engineering Company

R. M. P. Hamilton
General Engineering Company, Toronto, Ont., Canada

AN EFFICIENT METHOD of recovering inexpensively the cyanide and precious metal (if any) in the barren solution fouled by contact with cyanide-consuming ores has been developed by the patentees of the Geco process. Local conditions, of course, would govern the detailed application, but reference may be had to the accompanying flowsheet for a general case.

Pregnant or barren solution is run into one of three receiving tanks having a storage capacity equivalent to two hours' flow of solution, or it can be treated continuously. If the cycle method is employed, the first tank will be allowed to fill, then the second, and later the third. When the first receiver is filled, titrations will be made to determine cyanide and lime content per ton of solution. Reference to a table such as is shown herewith indicates an equivalent quantity of zinc sulphate added in the form of a 10 per cent solution, or other convenient strength. After about an hour of contact, the resultant precipitate will have settled in white, easily filtered floccules in the lower half of the tank. The pulp is then run through a standard Genter (filtering) thickener, which operates continuously from the discharge of one or other of the three receiving tanks. The Genter delivers a clear filtrate and a sludge product, the latter amounting to 2 or 3 per cent of the original solution. The sludge is stored in a small receiving tank for subsequent regeneration of cyanide, the plant to accomplish which is operated on day shift only.


Feed Solution (filtrate from cyanide gold-silver extraction plant)
Two zinc sulphate stock tanks replenished from regenerated zinc sulphate solution filtered (standard filter press) from regenerator residues
Three treatment tanks, each having stirring mechanism for mixing in zinc sulphate solution during two hour (maximum) treatment
Genter thickener, operating continuously on treated solution pulp from one or other of treatment tanks
Filtrate
Sludge
Acid
Zinc dust
Standard precipitation press
10% sulphuric acid
Milk of lime
HCN gas to cyanide absorption system
Regenerator
Gold-silver precipitate for refinery
Barren solution free from fouling constituents
Regenerator residues
Filter press
Regenerated lime-cyanide solution for re-use
Regenerator residues for smelting or special treatment
Regenerated zinc sulphate for re-use

Flowsheet, precipitation and cyanide recovery plant

The Genter filtrate, in a decyanided condition, is neutral, free from fouling constituents, and contains only the cyanide combined with the gold or silver not previously precipitated. If sulphocyanides occur, they can be precipitated by copper sulphate, sulphur dioxide, or other cheap reagent. To recover the precious metals, it is necessary to add to the decyanided solution enough acid to dissolve the weight of zinc chemically required to precipitate whatever quantity of gold and silver is present. The solution is then contacted with zinc, without evacuating or deoxygenating, and the precious metals are recovered in the form of a clean, high-grade precipitate. Zinc dust may be used with a standard type of filter press, or zinc shavings, according to local conditions. After contacting with zinc, these solutions may be regarded as "water," re-used in the mill, or run to waste if water is plentiful.

The Genter sludge goes to the regenerator—a lead-lined chamber having natural or forced draft connections to the absorber for hydrocyanic acid. This is a contact chamber for fixing the regenerated gases in lime water, for re-use in the mill.

When the regenerator is from one-third to one-half full of Genter sludge, sulphuric acid is added in proportion to the amount of cyanide and lime (or zinc sulphate) indicated by titrating the feed solutions, and by reference to the table shown herewith. Regeneration is aided by agitation for a period up to two hours. Samples of the regenerator solution will then be slightly acid to methyl orange. The cyanide, originally titrated in the feed solution and precipitated by zinc sulphate, will have been driven off from the regenerator charge as hydrocyanic acid gas and "fixed" as calcium cyanide ($Ca(CN)_2$)

Precipitation and Cyanide Recovery Process Chemical Equivalents

	2KCN	*2NaCN*	*Ca(CN)₂*	*Zn(Cn)*	*2KCN*	*CaO*	*H₂SO₄*	*Ca(OH)₂*	*ZnSO₄ 7H₂O*
*KCN**.....	1.	0.753	0.706	0.903	0.416	0.43	0.753	0.57	2.21
*CaO**.....	2.32	1.75	1.64	2.1	0.965	1.00	1.75	1.32	5.14
NaCn.....	1.328	1.00	0.94	1.2	0.552	0.571	1.0	0.756	2.94
2NaOH.....	1.62	1.222	1.15	1.47	0.675	0.7	1.222	0.925	3.6
ZnSO₄ 7H2O.....	0.453	0.342	0.32	0.408	0.188	0.195	0.342	0.2575	1.0
Zn.....	1.990	1.50	1.4	1.8	0.830	0.857	1.5	1.13	4.4
Ca(CN)₂.....	1.416	1.066	1.0	....	....	....	1.066	....	3.129

* *KCN* and *CaO* in solution are precipitated by their respective equivalent quantities of $ZnSO_4 . 7H_2O$.

CuCNS precipitation: 249.7 $CuSO_4 . 5H_2O$ + 248.2 $Na_2S_2O_3 . 5H_2O$ (mixed) in equal proportions precipitate *NaCNS*, *Cu*, etc.

Solubility of *CaO* in water is 0.14 per cent at 0 deg. C, and 0.07 per cent at 80 deg. C.
Solubility of $CaSO_4$ in water is about 1 part in 400, by weight.

by the lime water in the absorber. The zinc sulphate originally consumed in precipitating the cyanide and the fouling constituents in the feed will have been regenerated by the action of the sulphuric acid, so the residual sludge is filtered to recover the regenerated zinc sulphate solution. The resultant filter cake, containing base-metal constituents such as copper and iron, may be handled according to commercial valuation.

The process is cyclical, its net chemical requirement being the sulphuric acid needed to regenerate the hydrocyanic acid from the zinc cyanide precipitate, plus the small amount necessary for gold and silver precipitation with zinc, and the lime required to "fix" the regenerated hydrocyanic acid in lime water. No white precipitate can be formed to block solution flow; only the actual regeneration requires special attention, and this operation is carried out on day shift. Vacuum precipitation is unnecessary, and the precious-metal precipitate is clean and high grade. Over 95 per cent of the solutions going to the recovery plant may be re-used as fresh water in the mill circuits, which are thus kept active without the wastefulness of careful and troublesome handling of bulky solutions, which cannot otherwise be avoided except by sending fouled solutions to waste.

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Protecting Steel Tanks With Lead

A satisfactory method of protecting its many steel water tanks against corrosion has been developed by the Union Pacific Railroad System, according to the Lead Industries Association. The tanks are given a shop coat of ready-mixed red lead paint inside and out. After erection the exterior is given a brown and a black coat, both being mixtures of red lead and lampblack, with lampblack increased in the black coat.

Interiors receive three coats in addition to the shop coat. The first field coat is brown and is made by adding 10 oz. of lampblack paste, 6 fluid oz. of japan dryer, and 2 lb. of finely powdered litharge to 1 gal. of ready-mixed red lead paint. A second field coat, light brown, has the same composition as the first with the exception of the lampblack paste, 5 oz. of which is used instead of 10 oz. The third field coat, red, is the same, with all lampblack omitted.

The paint must be thoroughly mixed at the start and frequently stirred. A rather high proportion of pigment is desirable, especially on interior surfaces. Each coat must be brushed out to a thin film. Proper intervals of time must be allowed for the drying of each coat.

The litharge passes a No. 325 sieve with total residue on the sieve not exceeding 1 per cent by weight. The ready-mixed red lead-paint pigment contains 88 per cent of red lead by weight, which must run not less than 94 per cent true red lead. The lampblack paste is 25 per cent pure lampblack by weight, balance pure linseed oil. Addition of litharge gives an extraordinarily hard paint film that does not become unduly soft by continued soaking. The ready-mixed red lead paint contains 76 per cent by weight of pigment.

Tanks are inspected annually and painted at intervals of from four to ten years, depending upon local conditions. The tank is drained and the steel cleaned, sometimes by sandblasting, but more generally by scraping and wire brushes. Brush painting is usually used. The paints described cover about 400 sq.ft. per gal. with the brush method.

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Aluminum Booms for Draglines

Other Interesting Developments in Equipment

A major development of the year 1931 in the power shovel field was the new 18-yd. stripping shovel built by the Bucyrus Erie Company, South Milwaukee, Wis. This machine weighs over 1,000 tons and has thus far found its most extensive use in the strip mining of coal. Another development of unusual interest was the adoption of aluminum booms for dragline excavators.

Air Compressor of Mine Car Type

Having an over-all height above the rails of only 31 in., a new type of horizontal mine-car air compressor has been developed by the Worthington Pump & Machinery Corporation, Harrison, N. J. These compressor-cars are built with displacements of from 110 to 200 cu.ft., and will run one to three rock drills, depending on their size. The frame is made of heavy structural-steel channels and beams, electrically lock-welded. Front and rear bumpers are rounded to permit the car, when coupled, to negotiate curves without difficulty. The 110-cu.ft. car, fitted with a single, double-acting Feather Valve air compressor, weighs 6,250 lb. and has an over-all length of 115 inches. A 20-hp. motor (230-550-volt d.c. or 220-440-550-volt a.c. 60-25 cycles) drives the compressor through a V-belt drive.

New Electric Cap Lamps

Two companion lamps to the Edison Model H electric safety cap lamp have been introduced by Mine Safety Appliance Company, Pittsburgh, Pa. The new models are the Model J and Model K, both being officially approved by the U. S. Bureau of Mines and the Mines Department Testing Station of Great Britain. Model J weighs only 63 oz. and furnishes 26 candle power illumination. Its battery dimensions are $7\frac{1}{4}$x$3\frac{1}{4}$x $1\frac{1}{2}$ in. and it is the lightest-weight mine-lamp battery of its kind available today. Model K developes a maximum of 55 candle power and is said to surpass by far, in volume of illumination, any portable safety lamp ever offered to the mining industry. The latter lamp introduces a novelty in the design of electric cap lamps in that the battery employs three cells instead of the usual two. Both models have light-weight bakelite headpieces and double-filament highly efficient gas-filled bulbs. Both employ the Edison nickel-iron alkaline batteries, which are magnetically locked to prevent tampering.

A New Combination of Motor and Speed Reducer

A simple arrangement for powering various motor-driven equipment which operates at reduced motor speeds is afforded by new unit drives now available from the Westinghouse company's Nuttall Works, Pittsburgh, Pa. These drives, known as Gearmotors, consist of speed reducers combined with motors forming self-contained units only a few inches longer over all than motors alone. Advantages include economy in space requirements, easy and simplified installation, reduced number of drive parts, high efficiency of operation, and assured low maintenance requirements. Each unit includes a Westinghouse Type CS general-purpose induction motor and a double-reduction, non-planetary-type helical-gear speed reducer built onto the one-piece motor frame, forming a sturdy and rigid assembly which is supported entirely by the motor feet.

New Separator Works Successfully in Non-Metallics

A new magnetic separator of extra high intensity that makes separations heretofore practically impossible has been developed by the Dings Magnetic Separator Company, Milwaukee, Wis. With it weakly magnetic materials, such as slate, may be separated satisfactorily from gypsum or coal. It has worked successfully also on silica sand, feldspar, coal, gypsum, and other materials where they have become mixed with supposedly non-magnetic steel or abrasion. It removes biotite, muscovite, and pure oxide of iron from feldspar and silica and has demonstrated its value on many other combinations of materials that have always been stumbling blocks, where the alternative to leaving the objectionable ingredients unseparated has been to employ expensive acid treatment. The new machine operates on the induction principle, requires little horsepower, and has low current consumption.

NEWS OF THE INDUSTRY

UNITED STATES

Salt Lake City, Utah, Dec. 28

Eureka Bullion Ore Cut. Drifting on the 1,900 level from the Big Hill shaft, in the East Tintic district, by North Lily, International Smelting mining subsidiary, resulted during the latter part of December in development in Eureka Bullion ground of 2 ft. of ore in the quartzite formation. The ore assays 7 to 8 per cent copper, 30 oz. of silver, and $2.50 in gold per ton. The strike is considered important because the type of mineralization is characteristic of that found below the large silver-lead orebodies of the adjoining Tintic Standard mine, and also because the deposit lies near the main North Lily ore zone.

On the 1,250 level, a wide replacement zone was intersected in the limestone in running to connect with the Big Hill shaft. On the 1,080 level, the company is also drifting west to explore other territory recently acquired from Chief Consolidated. Information from reliable sources indicates that the Addie claims will be taken over by International Smelting and the driving of the drift from the Yankee shaft resumed.

Mammoth Acquires Gold Chain. The Mammoth Mining Company in December acquired the Gold Chain property, in the Tintic district, and development work is planned for this area, which has paid more than $1,000,000 in dividends. Operations have been resumed by Chief Consolidated Mining in its Apex Standard area.

North Standard Resumes. Officials of North Standard Mining announce that the company will resume prospecting on the 1,400 level of the East Tintic property after a three months' shutdown. The company will center attention on two strong showings on the lower level below a great cave struck above.

Miami, Ariz., Dec. 25

Arizona Unemployment Relief. Numerous curtailments of operations have caused acute unemployment problems in all Arizona copper-mining districts. Many Mexican families have been furnished with transportation to the international border, where their own country takes care of them; others have been given employment picking cotton in the state's agricultural districts, but many Americans and Mexicans in the mining camps are still without means of existence. To alleviate this condition the communities have organized relief associations, composed of committees of citizens, who supervise the spending of relief funds which are obtained chiefly from voluntary contributions of those still employed. The amount given in each case varies with the circumstances of the donor. Single men without dependents are expected to contribute monthly at least the equivalent of one shift's work.

In Miami the welfare association is spending about $2,000 per month for the benefit of the unemployed and indigent, and nearly all the money is spent for residents of the district. Those able to work are given employment on work of a public nature for which payment is made in orders on the commissary at which the association maintains a supply of coal, about a dozen staple food articles and donated clothing. Supplies are purchased at wholesale prices. The rent of the commissary building is donated as well as the cost of supervision and distribution. Indigents who were cared for by the county are now taken care of by the welfare association at a much lower cost to the taxpayers than formerly. Another result of curtailment of copper-mining operations is that tax payments by the copper companies and from other sources have been so reduced that city and county governments in the mining districts are having trouble in paying their way.

Hanover Mine Closed. On Jan. 1, the Hanover Bessemer Iron & Copper property at Fierro, N. M., is to be closed. This property, controlled by U. S. Smelting, Refining & Mining, has been a substantial producer of iron-copper ore for several years. Discovery of new ore at the Sunrise mine of Colorado Fuel & Iron is the principal reason for the closing of the Hanover mine, as C. F. & I. has purchased about 200,000 tons of Hanover iron ore annually. In addition, the drop in the price of copper has cut the company's byproduct revenue considerably, as it ships siliceous copper-iron ore to the El Paso smelter.

Houghton, Mich., Jan. 5

Curtailment in Copper Country. Mines in the Michigan district, the oldest important copper-producing area in the United States, are cutting production in conformance with the producers' agreement to restrict to 26½ per cent of capacity. On Jan. 1, Calumet & Hecla closed its Centennial and North Kearsarge mines, laid off all single men without dependents, and cut the number of shifts worked per man to two or three a week. Operations will be conducted on the day shift only, with the possible exception of the metallurgical units. The North Kearsarge unit has been producing through its No. 4 shaft. At Centennial, development work on the Kearsarge amygdaloid had been under way.

Copper Range closed its Baltic mine on Jan. 1, thus reducing output by 250,000 lb. monthly. Baltic still has some rich ground in the upper levels and may be reopened when metal prices improve. The Champion mine is now running on a four-day schedule.

Isle Royale, managed by Calumet & Hecla, has cut its workers to two or three shifts per week to reduce production to the desired quota. Mohawk, which has not been smelting concentrate, is not affected by the agreement and will not change its scale of operations, now limited to one shaft. Quincy and Seneca are closed.

San Francisco, Calif., Dec. 29

Vickery Brothers Build. Construction of a leaching plant for the recovery of gold from low-grade gossan deposits is now being undertaken at the Vickery Brothers property, near Kennett, Calif. The plant is similar to, although smaller than, that operated by Mountain Copper. It contains two leaching vats of about 100 tons' capacity.

Cherokee Plans Mill. Cherokee Gold Drift Mining is making plans for a 300-ton mill at its property, near Oroville, Calif. The new plant is expected to treat gravel more efficiently than can be done in sluices.

Alaska Juneau Output Cut. The handicap of bad weather again cut Alaska Juneau output in December, when the gross value of the output of the gold mine at Juneau, Alaska, was only $262,000, according to the company's monthly statement. During 1931, the total output was $3,740,500, of which more than $3,500,000 is gold. Silver and lead account for the remainder. Profit in December was $77,200; for the entire year, $1,521,550. The quarterly dividend has been increased from 10c. to 12.5c. a share.

New York, N. Y., Jan. 7

Copper Curtailment Plans. Domestic copper producers are now faced with the problem of cutting their production to 26½ per cent of normal, under the terms of the curtailment agreement adopted by leading producers last month. In many camps, this reduction will work a genuine hardship on the miners, many of whom have been working on reduced pay and part time for more than a year. Already, however, it is evident that the companies intend to live up to the agreement, which provides that the curtailment remain in effect throughout 1932 unless the price of copper reach 12c. a pound or unless total stocks of copper are reduced to less than the total deliveries in the preceding four months.

Nearly all the leading producers are joining in the curtailment, as more than 90 per cent of world copper output was represented at the meeting. Details of the reductions that will be made by some producers have been reported by our various correspondents and can be found in the accompanying columns. As the A.B.M.S. statistics have not been issued during the last two months, present world production of copper is not definitely known, but it is believed to be about 125,000 tons a month. Curtail-

ment should reduce the monthly output to 85,000 tons.

Phelps Dodge Announces Cut. Phelps Dodge Corporation, leading copper producer in the Southwest, has announced that it will reduce its copper output from 17,500,000 lb. of copper monthly to 8,500,000 lb. monthly. Under the new plan, the mines at Bisbee, including those recently acquired from Calumet & Arizona, will produce 4,000,000 lb. a month; the Morenci unit will produce less than 2,500,000 lb. a month; and the New Cornelia unit will produce 2,500,000 lb. monthly. In addition, New Cornelia will close during July and August.

Vertique Marble Drilling. Diamond drilling at its quarry near Ishpeming, Mich., has been started by the Vertique Marble Company. Sullivan Machinery Company, of Chicago, has been awarded the drilling contract.

New Mexico Rock Asphalt. Development of a rock-asphalt deposit, near Santa Rosa, N. M., has recently been started by the New Mexico Construction Company, under the supervision of V. K. Jones. Drilling has indicated extensive reserves. The Bureau of Mines and Mineral Resources of the New Mexico School of Mines has also made a study of the deposit.

Relief for Needy Engineers. Jobs have been found for nearly 50 per cent of the 1,200 unemployed engineers in the New York area who have registered at the Work Bureau in the Engineering Societies' Building, 29 West 39th St. Cash contributions of nearly $70,000 have been received by the Professional Engineers' Committee on Unemployed, in addition to pledges of several thousands. The Committee desires to raise $200,000.

Butte, Mont., Dec. 30

Anaconda to Close Tramway. Anaconda Copper has announced the temporary closing of its Tramway mine, effective Dec. 31. James R. Hobbins, vice-president, stated that the closing was caused by low market demand for copper. The men who have been working at Tramway will, as far as possible, be transferred to other mines. This will mean that after Jan. 1 most men will be on a half-time basis, working two weeks out of every four. Mr. Hobbins added that Anaconda hopes to be able to maintain the curtailed rate of production during 1932, if business is no worse than in the last half of 1931. Anaconda was one of the first companies in the United States to undertake rotation of employment among its regular employees on a part-time basis.

Weather Hampers Placers. The early advent of severe cold this winter has hampered placer operations in Montana's gold camps. At Virginia City, California-Alder Gulch had already started surface skimming with its electrically driven combination dragline and shovel when the cold weather came. The machine is ready to start operations early in 1932. In the meanwhile, the company is completing a power line to its Bell quartz mines. Shipments of gold ore from this property to the Anaconda smelter are expected to start in February. At Gold Creek, Mont., Gold Creek Mining had not finished its new 6,000-cu.yd. dredge. This will be completed in the spring to enable operations on the large gravel deposits available.

At Maggie Gulch, Big Belt Gold Mining expects to continue placer mining all winter, as its gravel bodies are tapped by shafts 60 to 100 ft. deep. The backers of the company have acquired the Buckeye gold group, near Jefferson City, and expect to continue development all winter. A mill has been installed and tests of the ore are being made.

I. B. Mining Opens Ore. After working six months in the old Excelsior property, near Bannack, Mont., I. B. Mining has opened ore that is believed to be part of the bodies formerly worked. The new fine-grinding cyanide unit of the company has been treating more than 50 tons of ore daily from the old Bannack workings and the Thompson tunnel. Results have been very satisfactory, according to I. B. Haviland, general manager. This property was financed by the Wrigley interests, of Chicago.

Shamrock Mill Complete. About 20 miles west of Avon, Mont., Shamrock Mines is just completing a 50-ton mill to treat gold ore by amalgamation and flotation. The company has built a complete camp and has cleaned out two tunnels. Production is expected to start at the end of January. A "jig-back" tramway connects the mine and mill. The property was financed in Canada by E. J. Hubbert, prominent Great Falls oil and gas man.

Basin Montana Plans. Shares of series "A" of Basin Montana Tunnel have been listed on the New York Produce Exchange. This company has virtually completed plans for the new 16,000-ft. tunnel at its property, near Basin, Mont., which has produced some copper, zinc, lead, and silver. Driving is expected to start early in 1932. The Grey Eagle shaft will be deepened from 600 to 1,000 ft. About 50 veins are expected to be cut at depths of from 600 to 1,600 ft.

Chloride, Ariz., Dec. 26

Chloride Con. to Add Unit. Chloride Consolidated Mines is making arrangements to install additional equipment for fine grinding, with subsequent cyanidation, at its Dixie Queen property, north of the Tom Reed mine, Mohave County, Ariz. At present, the company is recovering about $18 in gold per ton of ore treated by amalgamation, according to R. E. Butler, manager of the company, but loss in the tailing is considerable. The silver-lead-zinc properties are idle because of low metal prices.

Reno, Nev., Dec. 28

Treadwell Yukon Abandons Bodie. The attempt of Treadwell Yukon to develop a large gold producer at Bodie, Calif., has evidently ended unsuccessfully, for the company has suspended all operations, according to J. R. Duncan. Work on the underground vein deposits was suspended some time ago. The surface dumps and low-grade areas, expected to average about $1.50 to $2 in gold per ton, actually yielded only about 60c. per ton in the 400-ton test plant that was built to treat low-grade ore. In the spring, the equipment from this unit will be dismantled and moved to some other location.

When, early in 1930, Treadwell Yukon acquired working rights on almost the entire Bodie area, which had been productive in the 'eighties, and proceeded to unwater the old mines, ambitious hopes were held for the enterprise. Only one small high-grade oreshoot was opened in more than 20,000 ft. of underground exploration. The high cost of pumping, coupled with exhaustion of the one profitable vein, led to suspension of this underground work. On the surface, the company had hoped to block out as much as 50,000,000 tons of low-grade that might be extracted by power shovel and treated by a cheap flotation process.

Castle Peak Closes. Castle Peak Quicksilver has again been forced to close its mine and plant, near Virgina City, Nev. Cold weather and snow-blocked roads forced the suspension. Just before the recent storms, 70 flasks of quicksilver was shipped to San Francisco for storage. Castle Peak has opened new ore at depth.

Mining Company Dividends for December, 1931

Companies in the United States	Situation	Per Share	Total Disbursements
American Smelting & Refining, 6% pfd	Various	1.50 Q	262,500
American Smelting & Refining, 7% pfd	Various	1.75 Q	875,000
Eureka Standard, c, s	Utah	0.03 Q	45,000
Freeport Texas Sulphur	Texas	0.75 Q	547,337
Golden Cycle, g	Colorado	0.40 Q	66,000
Hecla Mining, l, s	Idaho	0.10 Q	100,000
Homestake Mining, g	S. Dakota	0.65 M	163,254
Mayflower Associates, h	Various	0.50 Q	136,500
National Lead	Various	1.50 QX	464,747
National Lead, pfd. "A"	Various	1.75 Q	426,433
Nevada Consolidated Copper	Nev., Ariz., N. M.	0.10 Q	485,725
St. Joseph Lead	Various	0.25 Q	487,882
Sunshine Mining, s	Idaho	0.02 I	30,000
Texas Gulf Sulphur	Texas	0.75 Q	1,905,000
Tintic Standard Mining, s, l	Utah	0.10 Q	115,332
Utah Copper	Utah	1.00 Q	1,624,490
Companies in Other Countries			
Hollinger Consolidated Gold	Ontario	0.15 FWX	738,000
International Nickel, n, c	Ontario	0.05 Q	729,200
Lake Shore, g	Ontario	1.00 QX	2,000,000
McIntyre-Porcupine	Ontario	0.25 Q	199,500
New York & Honduras Rosario, g, s	Honduras	0.50 X	100,000
Noranda Mines, c, g	Quebec	0.50 I	1,119,886
Sylvanite, g	Ontario	0.02 SA	65,990
Total			$12,591,290

s, silver; l, lead; c, copper; n, nickel; g, gold; q, quicksilver; h, holding company; Q, quarterly; M, monthly; X, plus extra; FWX, two four-weeks' dividends plus extra payable in Canadian funds; SA, semi-annual; I, irregular

Coppermines Studies Geology From Air. Consolidated Coppermines, second largest copper producer in Nevada, has been studying the geology of its holdings near Ely, Nev., from the air. E. N. Pennebaker, geologist, took a series of air photos from an altitude of 6,000 ft. by which he could co-relate the various formations. In a talk at the Mackay School of Mines, J. B. Haffner, manager of Coppermines, stated that before the advent of Mr. Pennebaker as geologist, the company had believed most of its ore was in the zone of secondary enrichment. Drilling has substantiated Mr. Pennebaker's theory that the richest ore is in the primary zone.

New Company at Scossa. J. & C. Scossa Mines, Inc., of which Charles Scossa, one of the original discoverers of the gold deposits at Scossa, Nev., is president, has been formed to operate 70 acres to the south of Dawes Gold Mines, holder of the original discovery claims. The new company is being financed by Charles Hedwall, Jr., of Minneapolis, Minn. A two-compartment shaft is to be put down midway between two parallel veins, 300 ft. apart. Work on one of these veins, at a depth of 50 ft., indicates a small high-grade stringer.

Silverton, Colo., Jan. 2

Shenandoah-Dives Tonnage Up. During 1931, Shenandoah-Dives Mining, operating the Mayflower copper-lead-gold property here, mined about 190,000 tons of ore. The mill, originally built as a 300-ton unit, actually handled 171,000 tons, from which 8,086 tons of concentrate was recovered. At the end of the year, about 550 tons was being handled daily. A raise is now being put up to connect the Mayflower workings with the Shenandoah claims, which have not furnished any mill tonnage for the current operation.

Spokane, Wash., Dec. 28

Bell Boy Plans Mill. A. W. Fahrenwald, metallurgist, is preparing plans for a 150-ton flotation mill to treat gold-lead-silver ore from the Bell Boy Mining property, at Marysville, Mont. M. J. Savage is president and manager of the company, which holds four groups of claims. The Tousley tunnel has been driven into Bell Boy ground, 200 ft. below the old workings, and shows ore which, Mr. Savage says, averages 3 per cent lead, 2 oz. of silver, and $8 to $10 in gold per ton. Samples of the ore are now being tested.

Montana to Reopen Mine. Montana Mines will reopen the Spring Hill mine and mill, near Helena, Mont., according to Gust Carlson, vice-president of the company. About 60 men will be employed in the production and treatment of 350 tons of ore daily. Grade is about $6 in gold per ton. The mine was closed down several months ago because of difficulties with the owners. E. E. Jordan & Company has been appointed fiscal agents of Montana Mines.

Northwest Phosphate Developments. A new grinding plant, with a capacity of 100 tons of phosphate rock daily, is being built at Georgetown, Idaho, by Phosphate Fertilizer, of Spokane. Oscar F. Nordquist, of Wallace, Idaho, has charge of the operations. A fine-grinding plant will also be erected in the near future in the Los Angeles district, according to W. H. Honefinger, manager of the company.

In Montana a new phosphate field has been developed by Pacific Phosphate & Chemical, according to Allain Swain, president of the company. Open-pit mining will start on the holdings, which are near Elliston, Mont.

Unwater Gold Hill & Iowa. Restoration of the electric power line to the Gold Hill & Iowa mine, operated by Talache Mines, at Quartzburg, Idaho, has permitted unwatering the property and cleaning out the workings to the 900 level. Last summer a forest fire completely destroyed surface workings at this property, one of the largest gold producers in Idaho. Recent work at 700 and 900 ft. had opened ore of good grade. The shaft is bottomed at 1,100 ft. and the management intends resuming work there.

Denver, Colo., Dec. 27

Vanadium Buys Naturita Mines. Vanadium Corporation of America has foreclosed the property of Rare Metals Corporation, at Naturita, Montrose County, Colo. Vanadium Corporation was the only bidder at a sheriff's sale on Dec. 21, according to a report filed in the federal court here. Its bid was $428,000, which is $4,524 less than the sum it claims is due it on mortgages. Rare Metals borrowed the money to develop the property, which carries vanadium ores. A mill was built in 1930, but operations were suspended in October.

Tulsa, Okla., Dec. 28

Geological Society Now Wealthy. The Geological Society of America, meeting here for its annual sessions, reports that it has received about $4,225,000 through the will of R. A. F. Penrose, former president of the organization, who died on July 31 last. Mr. Penrose left an equal sum to the American Philosophical Society. What disposition will be made of the income—which will approximate $200,000 annually—is not known.

Washington, D. C., Dec. 28

U. S. Gold Output Rises. Production of gold in the United States, including the Philippine Islands, was 2,365,881 oz. in 1931, according to an estimate by the U. S. Bureau of the Mint. In 1930, United States gold production was 2,283,603 oz. If the Philippine Islands are not included, 1931 output was 2,191,881 oz., compared with 2,100,395 oz. in 1930. This increase of nearly 100,000 oz. in United States output was something of a surprise as the big drop in base-metal production had been expected to lessen byproduct gold recovery to counterbalance increased activities in the gold camps. California output, with a gain of 60,000 oz. to 510,235 oz., showed the biggest individual increase among the states. South Dakota, Alaska, and Colorado also made marked gains. Utah and Arizona, both base-metal producing states, showed substantial losses. Montana and Nevada totals were virtually unchanged. A drop of 11,000 oz. in Philippine output to 174,000 oz. was surprising. Silver production dropped from 50,748,127 oz. to 30,967,618 oz., every important state showing a lessened output.

House Silver Inquiry. A thorough investigation of the silver situation will be undertaken by the House Committee on Coinage, Weights and Measures, according to the plans of the chairman of the committee, Representative Andrew L. Somers, of New York. Representative Somers is a member of the Tammany delegation. A resolution authorizing the committee to make a special study of the silver situation already has been introduced. Usually, Congressional committees conduct hearings on a specific bill. In this case, however, the purpose is to defer the drafting of a bill until after a study has been made.

Dill Proposes Fume Payments. For the relief of citizens of the State of Washington who complain of damage from the fumes of Consolidated Mining & Smelting at Trail, B. C., a bill has been introduced by Senator Dill, of Washington, authorizing the Secretary of State to pay damage claims. Senator Dill contends that the fumes from the smelter have had a disastrous effect on the growing crops, orchards, timber, and farm lands of residents of the State of Washington living along the Columbia River from the Canadian boundary line southward and beyond Marcus.

Senator Dill also contends that the citizens who have suffered damages by reason of the fumes have repeatedly asked for relief from the State Department because of their inability to bring suit in the courts of Canada. Although the matter was referred to the International Joint Commission for settlement, final action was impossible because of the inability of the American members of the Commission to induce the Canadian members to agree on the amount of damages, Senator Dill stated. Under the terms of the bill submitted by Senator Dill, the Secretary of State would be authorized to pay all claims to a total amount not exceeding $1,500,000.

E.&M.J. Weighted Index of Non-Ferrous Metal Prices

100 Is Composite for 1922-3-4

1920	130.92
1921	80.80
1922	93.94
1923	103.43
1924	102.63
1925	111.74
1926	107.74
1927	97.54
1928	99.78
1929	110.33
1930	82.87
1931	60.20

	1929	1930	1931
January	106.86	102.94	67.62
February	110.37	102.28	65.90
March	122.71	99.93	68.03
April	116.36	93.24	64.45
May	110.03	83.41	59.81
June	109.66	78.57	58.23
July	109.58	74.58	59.54
August	109.81	74.41	58.05
September	109.29	73.42	57.09
October	108.52	69.63	55.08
November	106.09	71.39	54.91
December	104.75	70.65	53.77

El Paso, Tex., Jan. 5.

Reductions at El Paso. As part of the general movement of the copper industry to curtail production for the purpose of bringing about an improvement of the price of the metal, Nichols Copper will reduce the monthly output of its refinery here from 8,000 tons of refined wirebars to 3,000 tons, according to John P. Dyer, superintendent. The company has decreased its working force in the plant from 325 men to 175. The refinery will receive from the old Calumet & Arizona furnace in Douglas, Ariz., 6,500,000 lb. of copper a month, instead of the usual 17,000,000 to 18,000,000.

At the American Smelting & Refining smelter here, production will be cut in half by a system whereby employees will work a month and lay off a month, beginning Feb. 1. Of the 375 men employed, 75 will be retained in the off months, Brent Rickard, general manager, said.

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CANADA

Vancouver, B. C., Dec. 24

Georgia River Development. Encouraging development results are reported from the operation of the Georgia River Gold Mines, formerly known as the Guggenheim group, below Stewart, on the Portland Canal. The main vein is a silicified zone, 10 to 20 ft. wide, carrying some gold. In addition, several small rich parallel veins have been discovered that would prove valuable sweeteners if sufficient mill-grade ore can be developed. Tunneling is in progress at two points.

Premier to Drill Salmon. An active campaign of diamond drilling will be resumed on the Salmon Gold group, in the Portland Canal district, under bond to Premier Gold. Several highly mineralized silicified zones have been exposed in surface workings by the owners. One zone has a maximum width of 25 ft., including 9 ft. of massive pyrrhotite. Another zone is 7 ft. wide.

Additional ore has been opened in the Premier mine by the development work done in the upper levels of this famous silver-gold producer, which has been nearing exhaustion. The new ore is parallel to the main ore zone. Premier Gold, faced by the possibility that its original mine may soon end its career, has been extremely active in searching for new properties.

Lorne Mill Progress. Construction of the 100-ton mill at the Lorne mine, adjoining Pioneer Gold, in the Bridge River district, is nearing completion. Only a limited tonnage of silver-gold ore has been proved by Bralorne Mines, the operating company, but extended development is to be undertaken. Geological study of the property has indicated several possibilities that will be investigated.

Reno Ore Lengthens. On the No. 4 level of Reno Gold Mines, at Salmo, 400 ft. of ore has been proved that averages $40 a ton over a width of 2 ft. The vein is now being developed through the No. 5 level, 270 ft. deeper. The good grade gold ore being developed in the sulphide section of the vein, below the zone of oxidation, is considered a good augur for continuation of ore at depth. About 45 men are employed under O. C. Thompson, manager. Production in November was valued at $28,005, of which $3,843 represents premium received because of depreciation in Canadian currency.

Alexandria Financed. Satisfactory arrangements have been completed for financing further development of the Alexandria Gold property, on Phillip's Arm. This property is ideally situated at sea level. An oreshoot 350 ft. long, 5 ft. wide, and averaging $8 in gold per ton, has been proved in tunnel workings. A shallow shaft has indicated that the ore goes down. Present plans are to sink the shaft to 800 ft. below sea level, a program that is fully justified by present showings, according to government reports. T. S. Davey is manager.

Perrier Producing. The first shipments of concentrate have been made from the Perrier Gold property, alongside the Great Northern Railroad track, near Nelson. British Columbia Explorers Syndicate has acquired this mine. The 2-ft. vein on which an incline shaft has been sunk is being developed by F. J. Sur. The concentrate shipment, which went to the Trail smelter, consisted of 90 tons.

Toronto, Ont., Jan. 4

Nipissing to Finance Beattie. By a vote of 506,648 shares to 212,587, Nipissing shareholders approved the proposed arrangement whereby Nipissing Mining joins with Ventures, Ltd., in the development and financing of the Beattie gold property in northwestern Quebec. H. Park, general manager of Nipissing, reported that after checking over the diamond-drill records and dealing with them in the most conservative manner, the estimated tonnage was 5,000,000, having an average value of $3 a ton in gold. This winter about 200 men will be engaged in building roads, constructing camps, and sinking shafts. Before the end of 1932, construction is expected to start on a first mill unit with a capacity of 1,200 to 1,500 tons a day. Total costs are estimated at $2.25 per ton.

Wright-Hargreaves Output. In spite of increased appropriation for development work, large expenditures for new plant and equipment, starting of a new service shaft, and increased dividend payments, Wright-Hargreaves, at Kirkland Lake, ended 1931 with substantial additions to its cash surplus. Production is now about $260,000 in gold a month, without exchange. Mill capacity has increased 780 tons a day as the result of installing a new crushing plant. Underground development continues favorable. The north vein has been cut at the 3,000 level, where it showed ore of good grade.

Howey Tonnage Rises. Production at Howey Gold, in northwestern Ontario, is about $100,000 a month in Canadian funds with exchange taken into consideration. This increased production results from sorting a larger tonnage at the surface. At present 900 tons a day is being mined, of which 150 tons is sorted out at the surface. Total costs, including interest on loans, run under $3 a ton milled and about $2.55 per ton mined. The loan account, owed to W. S. Cherry, has been reduced to $400,000, and further substantial payments will be made during the winter months when purchase of supplies is reduced to a minimum.

Noranda Curtailing. Noranda Mines, largest producer in Quebec, has agreed to cut copper production from 60 to 40 per cent of estimated capacity, subject to certain conditions, but has not joined with Copper Exporters, Inc., in its move to have production cut to 26½ per cent, according to a statement issued by J. Y. Murdoch, president of the company. The conditions under which Noranda made its offer to curtail production of the red metal in cooperation with other concerns seeking to build up copper prices were that the company's output of precious metals remain as at present, and that its gold production of slightly more than $5,000,000 be maintained. Subject to these conditions, Mr. Murdock said, the company was willing to choose such ores as would cut down the incidental production of copper without interfering with the precious-metal output.

Inco Not to Cut. No further reduction will be made in production of International Nickel at present, according to official statement from the company's headquarters at Copper Cliff. International Nickel is a party to the agreement of the copper producers, but, in reducing its nickel output, it already has cut byproduct copper recovery to about the percentage stipulated in the agreement. International Nickel's copper producing capacity is close to 120,000 tons annually, indicating a present output of about 30,000 tons annually.

Treadwell Yukon Suspends. After a development campaign that lasted several years and that must have cost several million dollars, Treadwell Yukon has suspended work at its Errington property, in the Sudbury district. Efforts have been directed toward proving a large low-grade zinc-copper-lead deposit. Work on the upper levels had been successful in indicating a substantial tonnage. At 1,500 ft., the lowest level reached, mineralization had recently been encountered. Low prices of metals, however, forced discontinuance of the work. The 300-ton pilot flotation mill has been out of operation for more than a year.

No Curtailment in Manitoba. Hudson Bay Mining & Smelting and Sherritt Gordon, Manitoba's two new copper producers, are not making any cut in copper production for the present. These properties are now producing about 2,000 to 2,500 tons of copper monthly. In addition to copper, Hudson Bay is recovering zinc, gold, and silver from its Flin Flon ores and is largely responsible for the increase in Manitoba gold production in 1931 to 100,000 oz. It produced probably 70 per cent of that total. Operating costs for Flin Flon have not yet been published, but Sherritt Gordon costs are said to be down to 6.5c. a pound or thereabouts.

Ottawa, Ont., Jan. 1

Canada's 1931 Mine Output. Although production of Canadian mines in 1931 was valued at a lower total than in 1930, output of gold showed an increase of 27 per cent. In 1930, gold output was 2,102,068 oz. and in 1931 it was 2,679,728 oz., the increased production coming largely from the Kirkland Lake, Porcupine, Rouyn, and Flin Flon areas. Preliminary figures issued by the Dominion Bureau of Statistics put 1931 copper output at 290,581,122 lb., worth $23,772,000 (in 1930, 303,478,356 lb.); 1931 nickel output was 66,103,521 lb. (in 1930, 103,768,857 lb.); 1931 lead output was 267,850,972 lb. (in 1930, 332,894,163 lb.); 1931 zinc output, 236,861,153 lb. (in 1930, 267,643,505 lb.); 1931 silver output, 20,403,771 oz. (1930, 26,443,823 oz.); and asbestos, 162,278 tons (242,114 tons in 1930). The total valuation of 1931 output is estimated at $227,769,000. In 1930, mineral production was worth $279,873,578.

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SOUTH AMERICA

Paris, France, Dec. 1

Mocupia Plans Expansion. By the expenditure of 3,750,000 fr. (about $125,000), Compagnie Francaise de la Mocupia will be able to develop the Calvario and Colombia XI orebodies at its properties in Venezuela and increase milling capacity to 100-125 tons of ore daily. During the year ended July 31, 1931, production was 223 kg. of gold from 16,200 tons of ore, according to a statement made at the annual meeting. Cost of production will be reduced by the enlarged scale of operations to a point where ore carrying 8.5 grams of gold per ton can be treated profitably. Average grade is about 18 grams. The Calvario deposit would be developed through a new incline shaft, 100 m. deep.

La Paz, Bolivia, Dec. 2

Aramayo Mines Closed. Operations have recently been suspended at the property of Aramayo Mines, normally second largest tin producer in Bolivia. The reason for suspension is the exhaustion of high-grade silver ore, which has been treated during the last two years in preference to tin ore because of desire to curtail tin output. In addition, the company is remodeling the Atocha concentrator, which treats Chocaya ore, and it desires to conserve tin ore resources. In November, production was only 161 long tons of concentrate, compared with 189 tons in October. In 1930, the company sold 1,902 tons of tin concentrate, compared with 4,281 tons sold in 1929. Silver sales in 1930 were 3,353,435 oz. compared with 3,263,849 oz. in 1929. Reserves of high-grade silver ore at the end of 1930 were 55,114 metric tons, averaging 80 oz. of silver a ton and 3.63 per cent tin.

Tin Exports Down. Figures for the first ten months of 1931 show Bolivian exports of tin concentrate through West Coast ports totaled 42.571 metric tons of high-grade and 1,782 tons of low-grade. Corresponding figures for 1930 were 55,895 tons of high-grade and 4,827 tons of low-grade, indicating curtailment of about 25 per cent. Official Bolivian government figures for the five months from June to October show exports contained 13,026 metric tons of fine tin, of which Patiño Mines exported nearly 44 per cent, or 5,702 metric tons. La Unificada, of Potosi, controlled by the Rand Mines group, was the second largest exporter, with shipments totaling 865 tons.

Condeaqui Shut Down. The Condeaqui scheelite concentrator has been temporarily closed down, pending the arrival of the rainy season. This plant has been running for only a few months.

Washington, D. C., Dec. 18

Brazil Searches for Gold. The increased interest in gold has prompted the Brazilian Government to order its Geological Service to investigate domestic gold properties, according to U. S. Department of Commerce reports. At Lauras, Rio Grande do Sul, several deposits have been tested. A yield of 25 grams in gold per ton from some of these has been indicated if modern metallurgical treatment is used. Tests are being continued. The Service is planning exploitation of deposits at Assur, Bahia. Abandoned mines along the Central Railway, in Minas Geraes, are being studied. At Descoberto abandoned workings are being cleaned out.

New York, N. Y., Jan. 7

Cosach May Lead to General Strike. A general strike, to start Jan. 11, is reported to have been called by Chilean radical labor leaders. This strike is intended as a protest against Cosach, the giant nitrate monopoly, formed last year to acquire all the Chilean nitrate properties. Cosach's financial structure, devised by Anglo-Chilean Consolidated officials, has been under critical fire for some time. The radicals are asking for a dissolution of the monopoly, believing it to be unfair to the Chilean Government, which has a 50 per cent interest in it.

London, England, Dec. 15

Frontino Postpones Shaft Sinking. Sinking of a new shaft at the Silencio mine of Frontino Gold Mines, at La Salada, Colombia, has been postponed. Development work on the south end of the vein shows some remarkably good results on Nos. 19, 20, and 21 levels. As the new shaft was to have facilitated exploitation of ore below the No. 25 level, this new ore removes the necessity for going to greater depths at present. At the end of June, Silencio ore reserves were 81,000 tons averaging 17.2 dwt. in gold per ton, an increase of 21,000 tons in volume and 1 dwt. in grade over the previous year. The new Dona Teresa power plant permitted increases in both development driven and tonnage mined.

Frontino Gold acquired the Marmajito mine at the beginning of 1931. In the six months to June 30 development work totaled 1,034 ft. A new 750-cu.ft. compressor is being installed which will permit expansion of development. In the six months ore reserves were increased 6,000 tons to 26,700 tons averaging 20.3 dwt. in gold per ton. Production was 6,491 oz. of gold from 6,170 tons of ore. Preliminary work in connection with opening two new mines —Cordoba and Manzanillo—has been started.

Production Statistics

Recently issued annual reports have contained the following interesting data on metal production of the individual companies. Figures given are for the years ended June 30, 1931, and June 30, 1930, except where otherwise indicated:

	1930	1931
Fresnillo Company		**Mexico**
Oxide ore, short tons	825,876	780,200
Silver, oz.	3,771,033	2,831,811
Gold, oz.	3,948	3,624
Sulphide ore, tons	251,278	251,216
Lead concentrate, tons	35,427	38,350
Lead content, per cent	52.4	58.1
Zinc concentrate, tons	31,307	40,609
Zinc content, per cent	50.7	52.1
Mexican Corporation, Teziutlan		**Mexico**
Ore milled, short tons	63,760	65,764
Copper concentrate, tons	8,501	8,906
Copper content, per cent	20.22	20.0
Lead concentrate, tons	.....	757
Lead content, per cent	.....	47.6
Zinc concentrate, tons	9,805	10,932
Zinc content, per cent	57.5	57.6
Santa Gertrudis		**Mexico**
Ore milled, short tons	633,657	475,146
Silver, oz.	9,655,056	8,211,049
Gold, oz.	36,875	35,291
Frontino Gold		**Colombia**
Ore milled, long tons	24,380 (a)	36,140
Gold, oz.	18,491 (a)	37,130
Silver, oz.	(b)	25,356
Mount Lyell Mining (c)		**Australia**
Ore mined, long tons	244,535	292,830
Copper, tons	9,817	10,033
Silver, oz.	187,489	155,976
Gold, oz.	3,631	3,834
Trepca Mines (c)		**Yugoslavia**
Ore, metric tons	17,361	256,549
Lead concentrate, tons	2,229	41,358
Lead content, per cent	70	75.6
Zinc concentrate, tons	1,003	32,770
Zinc content, per cent	47.6	50.3
Pahang Consolidated (d)		**Malay**
Ore, long tons	186,050	174,100
Black tin, tons	2,526	2,799
Alluvial tin, tons	35	36
South American Copper (e)		**Venezuela**
Ore mined, long tons	.....	10,520
Copper concentrate, tons	.....	908
Copper content, per cent	.....	20
Weardale Lead (c)		**England**
Lead concentrate, long tons	3,785	1,421
Fluorspar, tons	6,895	6,577
Messina		**Transvaal**
Ore mined, long tons	276,207	313,383
Copper, tons	6,792	8,697
Sulphide Corporation		**Australia**
Ore mined, long tons	133,338	73,342
Lead concentrate, tons	24,907	12,971
Lead content, per cent	65.9	70.3
Zinc concentrate, tons	32,593	17,157
Zinc content, per cent	50.2	52.7
Malayan Tin		**Malay**
Ground dredged, cu.yd.	6,143,900	4,977,500
Tin concentrate, long tons	1,530	1,460
Southern Malayan Tin		**Malay**
Ground dredged, cu.yd.	8,845,700	7,776,500
Tin concentrate, long tons	2,018	2,075
Southern Perak		**Malay**
Ground dredged, cu.yd.	3,116,080	2,720,600
Tin concentrate, long tons	864	491
Mocupia, Francaise de la		**Venezuela**
Ore crushed, metric tons	.....	9,634
Gold, kg.	.....	124

(a) Frontino mine only, 1931 figures include Marmajito.

(b) Not available.

(c) For years ended Sept. 30, 1931, and Sept. 30, 1930.

(d) For the years ended July 31, 1930, and July 31 1931.

(e) For the year ended March 31, 1931.

AFRICA

Johannesburg, Transvaal, Dec. 29

Brakpan May Acquire Witpoort. Negotiations are now in progress for the acquisition by Brakpan Mines of the Witpoort Gold Areas property, which adjoins it on the southwest. Brakpan is the eighth largest gold producer on the Witwatersrand and one of the largest dividend payers. During 1931, it has milled about 100,000 tons of ore monthly, yielding over 7 dwt. in gold per ton, from its mine on the Far East Rand. It is controlled by Anglo-American Corporation.

Witpoort Gold Areas, in which General Mining & Finance is interested, consists of a total of 3,010 claims. Originally, this ground was worked for coal, but the extension of gold-bearing formations was recognized and some development was undertaken in the period 1908-1913. It was then closed until 1928, when it was reorganized. Development carried on since then has indicated about 365,000 tons of 6-dwt. ore. Exhaustion of its financial resources, however, caused suspension of development last month. The property can be developed through Brakpan No. 4 shaft, if the negotiations are brought to a satisfactory conclusion.

Daggafontein Starts Operations. Daggafontein Mines has completed construction of its new cyanide mill and has now started crushing operations. This is the second new mine to enter producing ranks on the Far East Rand within the last year. As the decision to build a mill at Daggafontein was announced only in November, 1930, and construction did not start until May, 1931, the speed with which the work was completed is remarkable. Daggafontein has a present capacity of 50,000 tons of ore monthly, but the plant has been built so as to permit increasing capacity to 70,000 tons monthly with a minimum of effort. The flowsheet is similar to that of the West Springs mill, the first to adopt all-sliming methods.

December Rand Dividends. Declaration of the December dividends for the Witwatersrand gold mines, all of which will be paid in February, included announcement of the resumption of payments by Randfontein and East Rand Proprietary, two large low-grade producers, which have not paid since 1925 and 1924, respectively. In both cases, the payment is 6d. per share. East Rand has 3,000,000 shares of 10s. par outstanding, indicating distribution of £75,000. Randfontein has 4,063,553 shares outstanding, indicating a payment of £101,588. In all, 25 mines will pay dividends. With the exception of Van Ryn Gold, they are making their payments in Union currency—that is, in pounds which are still at gold parity. Van Ryn is paying in depreciated English currency.

West Rand Developments. Developments on new areas in the Far West Rand, where Randfontein and West Rand Consolidated are now the only important producing mines, have been most encouraging. As a consequence, New Consolidated Gold Fields has secured further prospecting options on ground there. The company already had options on Middelvlei and Western Areas. Some drilling has been undertaken at Middelvlei No. 6.

Pouring gold at Boulder Perservance, one of the smaller producers of the Golden Mile, Kalgoorlie, Western Australia

De Beers Meeting. At the annual meeting of De Beers Consolidated, foremost factor in the world diamond market, held at Kimberley last week, Sir Ernest Oppenheimer, chairman of the board of directors, stated that the company was in a strong position despite the difficulties of the diamond trade. He said that the outlook for the trade is improving because of efforts on the part of large producers to restore confidence by curtailing output and selling on a really sound basis.

London, England, Dec. 24.

Rhodesian Copper Production. Within the last month, the progress of the plans for world-wide copper curtailment has necessitated a change in the programs of the leading Rhodesian copper companies. As is now generally realized, three operating companies control all the Rhodesian deposits that are likely to produce copper within the next two years. Of these companies, two are already producing—Roan Antelope and Rhokana. The third, Mufulira, has been constructing a concentrator at its Mufulira mine with the intention of shipping concentrate to the Roan smelter.

Because operation of all three plants at 26½ per cent of capacity would mean comparatively high operating costs, the companies have wisely made an agreement whereby Mufulira will not start its new mill for the present. Instead, its production quota will be allotted to Roan Antelope and Rhokana. Arrangements have been made to credit Mufulira with the profits realized on its share of production. As Mufulira is owned by Rhodesian Selection Trust (which is allied with Roan Antelope) and Rhokana, a mutually agreeable division was readily reached.

At the Roan Antelope meeting on Dec. 18, A. C. Beatty, chairman of the company, stated that capacity of the mill and smelter was 75,000 short tons of copper annually. The cost of producing copper, not including depreciation or debenture interest, is about £30 per long ton in depreciated English sterling. As Northern Rhodesia is on the sterling basis, Roan obtains the full advantage of the drop in the value of the pound. Just how severely Roan output will be cut remains to be seen, but 26½ per cent of proved capacity is about 20,000 tons a year, and that rate is certain to be exceeded. Roan ore reserves are unchanged at 108,000,000 tons of ore, averaging 3.3 per cent copper.

Rhokana started the first unit of its N'Kana mill on Dec. 11, according to Sir Auckland Geddes at the company meeting on Dec. 16. Smelter operation will start early in March. At the meeting of world copper producers, Rhokana was conceded an annual capacity of 150,000 tons of copper, on the basis of its 10,000-ton mill. Without including the Mufulira allotment, therefore, Rhokana's quota will be about 40,000 tons annually. Only the N'Kana mine will be operated, as its sulphide ore can be more easily handled than the higher-grade, more complex ore from N'Changa and N'Changa West. Total ore reserves of Rhokana are 270,780,000 tons averaging 4.3 per cent copper. This does not include the Bwana, King Edward, or Chifumpu mines, nor does it include the Mufulira reserves in which Rhokana has a 30 per cent interest.

Mufulira, which has reserves of 160,000,000 tons of 4.1 ore, originally planned to build a 6,000-ton mill to treat ore from its Mufulira and Chambishi mines. Later, because of the unsettled copper market, the company decided to build only one 1,500-ton unit. Work on this unit has now been suspended.

Messina Increases Reserves. In spite of extraction of 313,383 long tons of ore during the year ended June 30, 1931, Messina (Transvaal) Development, largest copper producer in the Union of South Africa, increased its ore reserves by nearly 100,000 tons to 1,241,379 tons. The company's output of copper in the fiscal year was 8,706 long tons, produced at an average cost of £39 per long ton. In the previous year, output was 6,800 tons costing £51 per ton. Development

of the "A" lode in the Harper mine continued to yield good results at depth. The company's report states that copper mineralization has been proved in the area (total holdings are more than 22,000 acres) surrounding the two mines, indicating reserves of medium-grade copper ore.

Bibiani Mill Suggested. J. S. Watkins, reporting on the Bibiani property, Gold Coast, states that sufficient gold ore has been developed by the year's work to warrant further prospecting and the consideration of the provision of treatment plant. He states that the mining will be cheap and that tests indicate that the ore is readily amenable to cyanide treatment. Installation of an initial 100-ton unit is suggested. On the No. 3 level, in the northeast end of the mine, the No. 12 crosscut encountered ore assaying 27 dwt. per ton over 25 ft. On the No. 4 level the two shafts were connected. Ore 400 ft. long and 27 ft. wide averaged 10 dwt. per ton. On the No. 5 level developments had been disappointing until recently, when in two crosscuts payable ore of fair width was encountered. Reserves are estimated at 180,000 to 200,000 tons of ore averaging between 8 and 10 dwt.

Brussels, Belgium, Dec. 15

Belgian Congo Gold Output. Paul Crokaert, Belgian Colonial Minister, stated on Dec. 8, in an interview with a local newspaper, that the Belgian Congo would probably produce 9,000 to 10,000 kg. of gold in 1933. In 1930, production was 6,539 kg. and estimates indicate that 1931 output exceeded 7,000 kg. (A kilogram equals about 32.15 oz. troy.) Kilo-Moto Gold, leading producer of the Belgian Congo, reports an output of 4,798 kg. in the first eleven months of 1931. This is about 200 kg. higher than the total output of 1930. Colonel Moulaert, chairman of the technical committee of Kilo-Moto, has sailed to investigate recent discoveries of gold in the eastern Congo in which he is interested.

Ship Congo Gold by Air. Last week the first shipment of Belgian Congo gold ever to be carried by airplane arrived at Antwerp. The shipment was made by the Imperial Airways service and took only eight days from the mine. When shipped out by railroad and boat the trip takes several weeks.

Belgian "Slave" Labor Denied. On Dec. 3, the Belgian Foreign Office instructed its ambassador in Washington to lodge a protest against the charges made in the United States that "slave" labor is being used in the Katanga copper mines. The frequent use of the word "slave" in connection with the natives employed by Union Minière that has characterized the copper-tariff discussions in the United States is provoking considerable indignation in Belgium and the Belgian Congo. This is particularly true because of the widespread feeling here that Union Minière has made a great sacrifice in curtailing its production of copper at present. Union Minière's costs are reported to be exceptionally low at present because of the high-grade Kipushi ores that are being handled, and operations are said to be on a profitable basis in spite of low copper prices. In fact, the decision to reduce output further has been attacked in the Congo newspapers, which charge that an increasing number of Belgian subjects are being deprived of work solely to maintain American living standards.

New Railroad Outlet for Katanga. A new outlet by railroad from the Katanga copper mines to the West Coast of Africa will be provided by construction of a railroad from Brazzaville, in French Equatorial Africa, to Leopoldville, and then on to Port Francqui. The new railroad that joins Pointe Noire and Brazzaville will be open to regular traffic in May, 1932. Governor General Antonetti of French Equatorial Africa has stated that the French Government will be willing to finance construction. At present, there is a railroad from the mines to Port Francqui, on the navigable Kasai River.

Geomines Equipping Properties. Geomines, which has important tin properties in the Belgian Congo, is centering its attention on equipping its alluvial areas for large-scale production. Shortage of colored labor and the low price of tin were factors which made the company decide to suspend production for the present, even though the management believes a profit could be made at present low prices. At Luvua, a 15,000-hp. hydro-electric plant is being built. A 90-km. transmission line will connect it with the properties. In addition, a reservoir to contain 10,000,000 cu.m. of water is being constructed on the Lukushi River. Reserves contained 100,000 metric tons of 70 per cent tin concentrate.

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ASIA

Dairen, Manchuria, Nov. 18

Japanese Metal Output. Production of metals in the Japanese Empire during the fiscal year ended July 31 is estimated as follows (figures for the previous year in parentheses): gold, 436,430 oz. (397.789); silver, 5,909,751 oz. (5,871,644); copper, 77,011 metric tons (69,267); and sulphur, 55,082 tons (71,230). Special effort has been made toward opening up new gold deposits, but no important discoveries have yet been reported. Geologically, the Hokkaido and Kyushu islands are promising areas for both placer and lode gold deposits, and work is being centered there.

Melbourne, Australia, Dec. 3

Malayan Tin Dividends. Despite the abnormally low price of tin, many tin-dredging companies are still paying dividends; Kramat Tin Dredging, Kampong Lanjut, and Asam Kumbang, three of the larger Australian-controlled companies in Malay, have recently announced dividends of 6d. per share. Satululo Tin, operating a dredge in Western Siam, has been able, during the financial year ended March 31, to increase its net profit from £13,783 to £14,650. The company treated 1,257,000 cu.yd. containing 14.8 oz. tin oxide per yard at a cost of 5.48d. per cu.yd.

Annual Reports of Mining Company Earnings

Company	Metals Produced	Location	Operating Profit	Net Profit	Net Profit Previous Year	Value Output	Dividends Paid
Camp Bird (*a*)	Holding...	Various.....	(*b*)	£67,161	£134,719	.....	£25,985
Con. African Selection (*a*)	Diamonds.	West Africa.	£152,560	£129,164	£207,209	.....	£37,467
De Beers Cons. (*a*)	Diamonds.	South Africa	(*b*)	£133,746	£2,126,905	£679,553 (*c*)	£400,000
Fresnillo (*a*)	Silver, lead	Mexico.....	$98,422	$104,141	$1,080,446	$2,775,448	.....
Frontino Gold (*a*)	Gold.....	Colombia...	£55,697	£37,368	(*d*)	£157,184	£7,688
Golden Horse Shoe (New) (*e*)	Gold.....	Australia...	(*b*)	£11,770	£2,609	£82,243 (*g*)	.....
Hoboken (*a*)	Refining...	Belgium....	79,800,163 fr.	62,013,726 fr.	(*b*)	.....	15,000,000 fr.
Idaho Maryland (*f*)	Gold.....	California...	$237,020	(*b*)	(*b*)	$574,573	(*b*)
Kamunting Tin (*a*)	Tin.....	Malay.....	(*b*)	£32,694	£22,774	£86,513	£18,750
Lake View & Star (*a*)	Gold.....	Australia...	(*b*)	£102,117	£38,890	£564,676 (*g*)	.....
Luipaards Vlei (*a*)	Gold.....	South Africa	(*b*)	£44,829	£37,033	£395,796	.....
Malayan Tin (*a*)	Tin.....	Malay.....	(*b*)	£36,661	£80,173	£105,578	£35,000
Messina (Transvaal) (*a*)	Copper....	South Africa	(*b*)	£29,145	£142,598	(*b*)	.....
Mocupia, Francaise de (*h*)	Gold.....	Venezuela..	521,257 fr.(*i*)	222,990 fr.(*i*)	(*j*)	2,089,565 fr.	.....
Mount Lyell (*e*)	Copper....	Australia...	£132,767	£78,056	£131,177	£587,615 (*k*)	£103,210
Pahang Consolidated (*l*)	Tin.....	Malay.....	£37,199	£35,228	£57,672	£218,322	£7,000
Santa Gertrudis (*a*)	Silver, gold	Mexico.....	£215,684	£205,091	£317,522	$3,751,825	£174,758
Southern Malayan (*a*)	Tin.....	Malay.....	(*b*)	£43,424	£96,678	£148,604	£32,855
Sulphide Corporation (*a*)	Lead, zinc.	Australia...	—£73,490	—£67,669	£72,593	£448,594	.....
Trepca Mines (*e*)	Lead, zinc.	Yugoslavia.	(*b*)	£109,825	£2,221	(*b*)	£55,000
Van Ryn Gold (*a*)	Gold.....	South Africa	£51,102	£54,958	£48,424	£500,348	£25,000

In general, the figures for operating profit (or loss) represent the balance after deducting cash expenditures from the value of production. The figures for net profit (or loss) include the addition of sundry income, and the deduction of charges for depreciation, taxes, or other extraordinary expenditures. Because of the difference in accounting methods of different countries and different companies, the figures are not strictly comparable.

(*a*) For the year ended June 30. (*b*) Not available. Some of this material is not published. In other cases, the companies have failed to send us their reports and we have had to depend on incomplete published summaries. (*c*) Represents sales, not output. (*d*) Company reorganized last year. (*e*) For the year ended Sept. 30. (*f*) For the year ended Oct. 31. (*g*) Does not include bonus received from Australian government. (*h*) For the year ended Dec. 31, 1930. (*i*) On the basis of the last six months of 1930 only, as production was not regular till then. (*j*) Company not producing in 1929. (*k*) Includes receipts from railroad operation. (*l*) For the year ended July 31.

PHILIPPINE ISLANDS

Manila, P. I., Jan. 3

Big Wedge Strikes Ore. Associated Press reports received here state that Big Wedge Mining, operating in the Baguio gold-mining district, has opened high-grade ore in a winze that has been sunk below the main tunnel. This tunnel cut the vein at a depth of 200 ft. below the surface 400 ft. from its portal. At last report, the winze was 60 ft. deep. In addition to the Antamok Central group, in which the strike was made, the company holds the Anaconda, Muyot, and Reynolds groups, a total of about 50 full claims. Antamok Central is east and north of Benguet Consolidated, which for a long time was the largest gold producer in the Islands. A. I. Reynolds is manager of Big Wedge.

Benguet Wins Suit. Suit brought by stockholders in Balatoc Mining against Benguet Consolidated has been decided in favor of Benquet. The suit concerned financing of Balatoc operations, now controlled by Benguet. Both companies operate properties in the Baguio gold district. The case will probably be appealed.

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EUROPE

Berlin, Germany, Dec. 22

Silesian Zinc Output Cut. Georg von Giesche's Erben, operating the German section of the Bleischarley mine, in Silesia, has reduced its working force from 1,250 to 725 as a result of additional curtailment of zinc production asked by the zinc cartel. Schlesische A.G. für Bergbau und Zinkhüttenbetrieb has also curtailed. Ore from the company's Fiedlers Glück mine is being sent to the Lipine smelter. Giesche Spolka, Polish zinc producer in which Anaconda is interested, has reduced its production of concentrate from the Bleischarley plant to about 7,000 metric tons monthly, as against 15,000 tons normally. Giesche would cut its output further if it did not have to pay interest on its American bond issue. Last year, the obligations totaled 19,000,000 reichmarks. Mines de Zinc de Guergour, French producer, has suspended production.

Soviet Aluminum Unit. Production from the new Russian aluminum smelter at Alpajewsk, in the Urals, is expected to start in March, 1932. The unit has a capacity of 30,000 tons of aluminum annually, but will eventually be increased to produce 75,000 tons annually. Russia has constructed two other aluminum units, with the assistance of Pechiney, French producer.

Italy and Aluminum Cartel. The Italian producers of raw aluminum, who have not been represented on the board of the Alliance Aluminum Company, of Basle, are reported to be considering acting independently of the Alliance, which is actually a cartel of European producers. The head of the Italian Board of Trade recently made statements which lend credence to this belief. Italian aluminum production has been growing, though under the technical guidance of experts from other countries.

German Silver Imports. Import figures show the effect of Germany's decision to increase the amount of its outstanding silver currency. In the first half of 1931, monthly imports of silver averaged 8,480 kg. In September, imports were 159,972 kg. and in October, imports were 175,054 kg. The total for the ten months is 457,748 kg., or triple the imports in the corresponding period of 1930. Mexico has been furnishing about 80 per cent of the imported silver. Yugoslavia is also increasing the amount of its silver currency.

London, England, Dec. 19

Trepca Enlarging Mill. Trepca Mines has placed orders for equipment to increase its mill capacity in Yugoslavia by 50 per cent. The additional unit is expected to be operating by August, 1932. Rated at 500 metric tons daily, the Trepca plant has actually averaged about 850 tons of ore daily. The annual report states that during the year to Sept. 30, 1931, 256,549 tons was treated, averaging 12.6 per cent lead and 7.6 per cent zinc. Recoveries averaged 96.64 per cent of the total lead and 84.52 per cent of the total zinc in the ore. Actual operating time was 95.7 per cent of the possible total.

Actual milling results, A. S. Howie, general manager, says, showed that the mine yielded more lead and less zinc than sampling of the blocks mined indicated. A re-estimation, as of Sept. 30, 1931, shows proved and partly proved reserves of 2,100,000 metric tons averaging 10.5 per cent lead and 7.8 per cent zinc. In this estimate no upward adjustment is included for the gain in lead, but zinc assays have been adjusted to a conservative figure based on mine sampling and actual extraction. Low-grade areas, not included in the foregoing estimate, may pay to mine if metal prices improve.

A method of recovering pyrite in the ore by flotation was developed, and investigations are being made with a view to marketing this product, which is high in sulphur. Total operating costs for the year, including all general, London, and shipping expenses, but not depreciation and taxation, were 18s. 8d. per ton of ore. Despite low metal prices, Trepca showed a net profit for the year of £109,825. After allowing for the dividend of 5 per cent, which absorbed £55,000, Trepca closed its fiscal year with £208,840 net surplus of cash assets.

Brussels, Belgium, Dec. 15

Italian Quicksilver. Production of quicksilver in the first ten months of 1931 by the Italian quicksilver mines totaled 988 metric tons, compared with 1,800 metric tons in the same period of 1930. Exports during the first nine months were valued at 18,486,000 lire, compared with exports worth 29,363,000 lire in the first nine months of 1930.

Dives Directors Ask Support. At the annual meeting of Electro-Metalurgie de Dives, held in Paris on Dec. 9, heated discussion arose concerning the management of the company. The president asked the support of the stockholders, saying that the company had assets of more than 300,000,000 fr. He admitted that errors in judgment had been made in the past, but said that the board of directors had mapped a program which would bring the company, operating the only copper refinery in France, in closer contact with the Cie. Generale de Electricité. A vote showed 80,464 shares in favor of the directors and 1,694 against. Most of the shares were not voted. The chief points of contention were the company's investments in mines, notably the Sulitelma, in Norway, in which 42,000,000 fr. was invested, and speculations in metal prices.

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AUSTRALIA

Kalgoorlie, W. A., Nov. 22

Associated Northern Suspends. Associated Northern Blocks will probably relinquish its interests in Western Australia at the end of this month, after 32 years of operation. The company took over the Iron Duke lease in 1899 and worked it until 1915. In 1912 it acquired the Gimlet leases at Ora Banda, which have since yielded more than £500,000 in gold, for £29,000. Lack of sufficient water has made large-scale operation impossible. The company paid its last dividend in 1920.

Sons of Gwalia Sinks. Sinking of the main shaft on the Sons of Gwalia property, at Mount Leonora, has been resumed. The shaft had been bottomed at 4,000 ft. [A London report states that a winze, sunk below the No. 24 level, shows ore 5 ft. wide assaying 40s. in gold per ton, or nearly double the mine average.] Diamond drilling on the property at South Lancefield, Laverton, over which Sons of Gwalia had an option, has been abandoned.

London, England, Dec. 23

Wiluna Reserves Re-estimated. Carl O. Lindberg, consulting engineer for New Consolidated Gold Fields, which has a considerable interest in the Wiluna Gold property, near Meekatharra, W. A., has re-estimated ore reserves at the property. The first six months of operation of the new mill had indicated that the original estimate of 1,400,000 tons averaging 39s. per ton in gold was subject to revision. The new estimate is 780,000 tons of positive ore, averaging 33s., and 125,000 tons of probable ore of the same grade. T. M. Owen, consulting metallurgist, recommends increasing plant capacity to 40,000 long tons of ore monthly by addition of another thickener, drag classifiers, additional flotation cells, and a new Edwards roaster.

Although the Wiluna plant, as it now stands, was designed to treat 40,000 tons monthly, it has actually handled about 25,000 tons. Trouble with the power plant has been another handicap to operations. In November, output was 26,065 tons of ore, from which £30,635 in gold was recovered. Including premium, the output was valued at £38,064. Diamond drilling on the 625 level indicated 3 ft. of good ore 110 ft. west of the West lode shear zone, an entirely new occurrence.

Sulphide Reserves Down. Ore reserves at the Sulphide Corporation property, at Broken Hill, New South Wales, dropped by the tonnage mined to 707,000 long tons at June 30, 1931. The only one of the company's operations still running is the Cockle Creek Works, where sulphuric acid and superphosphates are produced and zinc concentrate is roasted. During the six months of the fiscal year that the Broken Hill mine was running, 77,303 long tons of ore was mined and milled, production being 12,971 tons of 70 per cent lead concentrate and 17,157 tons of 53 per cent zinc concentrate.

Melbourne, Australia, Dec. 3

Broken Hill Mill Efficiency. Improved metallurgical recoveries are being maintained by the three operating companies in the Broken Hill district. During the four weeks ended Oct. 31, Broken Hill South treated 26,470 tons of ore, assaying 15.2 per cent lead, 13.8 per cent zinc, and 7.2 oz. of silver per ton. From this, 95.7 per cent of the lead content and 89.6 per cent of the silver was recovered in a lead concentrate averaging 74.9 per cent lead, 4.4 per cent zinc, and 33.1 oz. of silver. Zinc concentrate assayed 53 per cent zinc and contained 85 per cent of the zinc content. In November, the plant treated 25,960 tons averaging 14.4 per cent lead.

Block 14 May Reorganize. Broken Hill Block 14, one of the less important Broken Hill mines, incurred a loss of £9,120 for the half year ended Sept. 30, 1931. The mine has been idle since July, 1930, and there is little prospect of operations being resumed unless metal prices substantially increase. Cash resources of the company are now at a low level, one of the reasons being the burden of the Workers' Compensation Act, under which the actuary appointed by the N.S.W. government had assessed the company's future contribution to the Workers' Pneumoconiosis and Tuberculosis fund at £62,736, half of which the government has to bear. The committee has made a demand for security against the future contributions of £31,368 and the company has agreed to give a charge over the mine and plant. Plans are being formulated to reconstruct the company so that the mine may be kept in good order and be ready for production when metal prices improve.

Australian Radium Resumes. The Australian Radium Corporation has resumed operations at Mount Paynter, South Australia, and plans to forward to the treatment plant at Dry Creek, near Adelaide, a test parcel of high-grade ore. Stoping is being carried out at the 50 level, where the secondary radium-bearing minerals, autunite and torbernite, are showing freely in quartz-ilmenite-ironstone gangue.

Batavia River Placers. Along the Batavia River, in north Queensland, rich alluvial deposits have recently yielded excellent returns. During the four months ended November about 2,800 oz. of gold was won from four separate mines. The richest alluvial has been worked at a depth of 70 ft., but gold has been found down to 140 ft. The area cannot be worked during the wet season from January to May. Port Stewart, 120 miles away, is the nearest town to the field, and with no permanent means of transport, the cost of supplies is heavy.

Titaniferous Sands Tested. Attention is being directed to the production of titanium white pigment from the titaniferous black sands along the shore of King Island, near Tasmania. Titanium Products Pty., Ltd., has been formed, and preliminary investigation work is in hand. The sands are believed to contain 30 per cent titanium oxide, 15 per cent zirconium oxide, and less than 0.5 per cent tin. This attempt follows that of the Australian Radium Corporation, which unsuccessfully investigated the manufacture of titanium white from the radium-bearing ore at Mount Paynter, South Australia.

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MEXICO

Mexico City, Mexico, Dec. 28

El Tambor Declared Unsanitary. A black picture of conditions obtaining among gold hunters who flocked from all parts of Mexico to the El Tambor placer fields of San Ignacio municipality, Sinaloa, is drawn by Dr. Pedro Arzani, who was delegated by the Department of Public Health to investigate alarming reports concerning the zone. According to the investigator, disease, resulting from bad water and deplorable sanitary facilities, and sanguinary brawls, have decimated the prospectors. He declares that approximately 60 per cent of the host that sought gold at El Tambor has perished from disease or been killed in fights caused by disputes over claims and divisions of proceeds, but avers that bad water has accounted for most of the fatalities.

Gen. Macario Gaxiola, Governor of Sinaloa, has been urged by the Department of Public Health, a federal organization, to arrange for the provision of good water in San Ignacio municipality, in which the gold fields are situated, as if conditions continue as they are a terrible epidemic may spread throughout the state.

Dos Estrellas Loses Dispute. Under a ruling by conciliation board No. 15, sitting in Toluca, Compañia Minera "Dos Estrellas," operator of mines in the states of Mexico and Michoacan, must provide company stores for its workers. Employees complained to the board that the company had closed its store, which, they contended, was a violation of labor contracts. The board's ruling compels the company to sell on credit to workers, prohibits it from deducting such bills from wages, and allows employees to liquidate voluntarily such accounts. As the result of disputes with its workers, the company recently closed its stores in Michoacan. Workers charged profiteering. The company asserted that the stores were operated at cost and that increased wholesale traffic necessitated raising retail prices.

New Gold-Silver Taxes. New regulations governing the percentages of the worth of gold and silver the federal government will collect on the production of these metals during 1932 provide the following, the Ministry of Finance announces: Gold: native metal, 10.5 per cent; in concentrates, 10 per cent; in precipitates or impure or mixed in bars, 9.25 per cent; and refined, 9 per cent, when its official value per kilo is 1,333.33 pesos. Silver: native, 7.5 per cent; in concentrates, 7 per cent; in precipitates or in bars, 2.75 per cent, and refined, 2.50 per cent, when its official worth is 26.65 pesos the kilo.

London, England, Dec. 19

Teziutlan to Close. Application for permission to suspend operations has been made by Mexican Corporation with relation to its Teziutlan mine, Aurora, Puebla, Mexico, according to F. W. Baker, chairman of the company. Teziutlan, a copper-zinc unit, has been operating at a loss for some time, but it has continued because its important new Minerva orebody is under development. Now that it has substantial reserves blocked out in this body (reserves on June 30 were 159,535 short tons averaging 3.2 per cent copper, 3 per cent lead, 17.5 per cent zinc, 3.5 oz. of silver, and 0.87 dwt. in gold per ton), and that the prospect of a substantial rise in metal prices seems slight, continued operations are not justified. Costs at Teziutlan averaged $10.29 a ton in the fiscal year ended June 30, compared with $12.17 a ton in the previous year. Recently they have been cut further, but still are not sufficiently low to eliminate losses.

Fresnillo Development Cut. The reserve position at the Fresnillo property, Fresnillo, Zacatecas, seems so assured that the local management has decided, on grounds of economy, to curtail development, F. W. Baker, chairman of Mexican Corporation, which has a 50 per cent interest in Fresnillo, told the annual meeting of his company on Dec. 11. Reserves of sulphide ore at Fresnillo on June 30 were 1,386,700 tons averaging 8.7 per cent lead, 9.2 per cent zinc, 0.6 per cent copper, 9.1 oz. of silver, and 0.3 dwt. of gold per ton, an increase of 400,000 tons over the reserves on June 30, 1930. Oxide-ore reserves, carrying only gold and silver, are down to 781,531 tons, but this part of the company's Fresnillo operations is no longer of great importance.

Santa Gertrudis Exploration. About 1,000 ft. of drifting will have to be done in the western section of the Dos Carlos mine of Santa Gertrudis, at Pachuca, Hidalgo, to pick up the extension of the rich Casualidad vein, worked by Real del Monte. Most of the Dos Carlos exploration work is in this section of the property. Santa Gertrudis has now to depend on Dos Carlos for about 75 per cent of its ore tonnage, as the El Borde and Malinche mines have been closed. Reserves on June 30 were 515,829 tons, averaging 21.2 oz. of silver and 1.9 dwt. in gold per ton. Only 100,000 tons was added during the fiscal year, while extraction was 475,000 tons.

Santa Gertrudis is negotiating for a lease on the El Bote zinc-lead-silver property at Zacatecas. Development will be centered on the extension of the vein system at depth. If ore is found, a low freight rate can be obtained for shipping ore to Fresnillo, where it can be treated in the Fresnillo mill.

Copper Price Maintained at 7¼c. in Dull Market—Zinc Stocks Reduced

The turn of the year found producers and consumers of non-ferrous metals marking time, for sales in all of the major items fell to a very low level. However, there is nothing unusual about such a state of affairs. Most operators appeared to be satisfied to hold aloof until developments in the international economic situation take some definite form. Curtailment in production of copper, bringing the rate of operations down to the agreed 26½ per cent of estimated capacity, is being put into force promptly in virtually all directions. The first of the year also witnessed the scheduled further reduction in the output of tin. The current rate of operations in lead and zinc appears to be sufficient to curb the upward trend in stocks. Silver production continues to decline, which tends to lend some support to the market. Quicksilver was dull and available in lots of 100 flasks or more at $65 per flask.

Copper Steady at Advance

During the month of December the price of copper advanced from 6¼c., the all-time low, to 7¼c., delivered Connecticut basis. The uplift in price took place soon after it became known that producers had reached an agreement on curtailment in production as well as on the revision of the rules of Copper Exporters, Inc. World production of copper, effective Jan. 1, 1932, is expected to fall to less than 90,000 tons a month.

Total stocks of blister and refined copper in the hands of North and South American producers at the end of 1931 probably exceeded 700,000 tons, though no official statistics have been issued covering the last quarter of the year.

Sales of copper improved around the middle of the month, but tapered off to next to nothing toward the close of the year. Shipments to domestic consumers during November declined to about 25,000 tons, and it is probable that December shipments held at about the same low rate.

A general advance in freight rates went into effect Jan. 4. On copper this would amount to about 2c. per 100 lb. In view of the fact that the refiners now hold large stocks of copper shipped before Jan. 4 on which the rate will not apply, and that lower rates have been granted to at least one of the plants in the New York district, the differential of 0.225c. between the Connecticut Valley basis and the f.o.b. refinery price will be continued for the time being.

Domestic producers of copper are keeping the tariff question alive, but few appear to look for any action on this score from the present session of Congress. Some domestic producers, however, do feel that a tariff on copper may be necessary in two or three years and that the work done at this time will not be lost. The excess of domestic over foreign copper costs, according to the findings of the Tariff Commission, is 1.54c. when depletion and interest are included, and 2.26c. without these items.

Export sales during December amounted to 28,500 short tons. The export price settled at 7½c., c.i.f. usual European ports.

Copper Exporters, Inc., has announced that after an exchange of cables with Europe the principles under which the association will work, beginning Jan. 1, 1932, have been agreed upon and were approved by the Board of Directors on Jan. 7. At a subsequent meeting the revised rules of the export association were adopted by all of the members. The custom smelters, according to reports, will be given every opportunity to dispose of their intake in the export market, thus taking the selling pressure in dull periods off the domestic market. It is believed that about 12,000 tons of copper monthly represents the present custom intake.

Lead Quiet

Demand for lead was slow, but sufficient in volume to remove the pressure from those sellers anxious to dispose of their current intake. The market settled at 3.75c., New York, the contract basis of the American Smelting & Refining Company, and at 3.55c., St. Louis. About 38,000 tons of lead were sold in the domestic market for shipment during December, a gain over the two preceding months.

World lead statistics for November revealed an increase of 5,465 tons in production credited to Australia. This was largely responsible for the gain registered during the month in the daily rate of production that the American Bureau of Metal Statistics estimated at 3,980 tons, against 3,806 tons in October, and 3,654 tons in September. The Britannia Lead Company, England, began the production of refined lead from Mount Isa (Australia) bullion in November.

Moderate increases in output for November were reported by Mexico and Spain. Production in the United States and Germany declined.

Lead production statistics for September, October, and November, in short tons, follow:

	Sept.	Oct.	Nov.
United States	31,966	36,546	31,671
Canada	10,262	10,537	10,391
Mexico	16,629	17,886	18,806
Germany	9,736	10,567	8,057
Italy	2,157	2,679	2,617
Spain and Tunis (*a*)	6,715	6,336	7,549
Europe, n.e.s. (*b*)	15,600	13,900	15,600
Australia	8,266	10,135	15,837
Burma	6,698	6,698	6,670
Elsewhere (*b*)	1,600	2,700	2,200
Totals	109,629	117,984	119,398

(*a*) Partial. (*b*) Estimated or partly estimated.

Zinc Unsettled

Despite the improvement in the statistical position of zinc, the market was so quiet that prices remained unsettled. Scattered lots again sold at 3.125c., St. Louis, equaling the low for the year.

Current Statistics of Production and Stocks of Copper, Lead, and Zinc

Data from American Bureau of Metal Statistics, American Zinc Institute, and Metallgesellschaft

All Figures Except Tin Represent Tons of 2,000 Lb.

	1930	1931										
COPPER — North and South America	Dec.	Jan.	Feb.	March	April	May	June	July	Aug.	Sept	Oct.	Nov.
Production, refined, daily average	3,431	3,305	3,566	2,292	3,350	3,313	3,276	3,110	2,909	2,890	.....	.....
Domestic shipments	69,854	60,209	60,636	75,685	54,567	45,265	50,217	43,144	45,816	40,459	.....	.....
Foreign shipments	39,163	45,597	39,415	36,797	32,218	26,684	33,251	26,321	29,016	22,124	.....	.....
Stocks, blister and refined	585,974	574,464	566,853	553,016	561,797	589,245	600,827	620,075	631,880	658,321	.....	.....
LEAD — United States												
Production, refined, from domestic ore	48,517	43,405	39,464	41,775	35,498	39,519	30,708	32,157	34,144	31,966	36,546	31,671
Production, secondary and foreign	4,692	3,892	4,654	3,025	2,941	3,598	3,683	4,409	5,453	3,934	4,722	3,820
Production, total, daily rate	1,716	1,526	1,575	1,448	1,280	1,390	1,146	1,179	1,151	1,163	1,331	1,180
Shipments reported	40,595	37,633	34,439	36,761	35,324	34,081	37,054	42,219	38,590	38,059	34,276	31,216
Stocks, end of month	103,247	113,145	122,826	130,426	132,993	142,370	139,698	133,958	134,977	132,804	139,796	144,057
ZINC — United States												
Production, daily average	1,054	1,049	1,056	1,043	971	971	783	689	692	712	699	684
Domestic shipments	34,245	31,022	30,249	35,224	27,418	25,831	27,604	28,460	23,599	20,902	21,163	20,327
Export shipments	.....	.....	.....	.....	.....	20	.....	20	.....	.....	.....	.....
Stocks, end of month	143,576	145,176	144,389	141,493	143,212	143,049	138,928	131,833	129,701	130,155	130,666	130,865
World Production Rate (Daily Average)												
Copper	4,395	4,174	4,596	4,418	4,296	4,209	4,224	3,919	3,924	4,023	.....	.....
Lead	4,923	4,694	4,883	4,693	4,898	3,988	3,995	3,691	3,906	3,654	3,806	3,980
Zinc	(*b*) 3,490	3,678	3,721	3,574	3,429	3,212	3,127	2,860	2,766	2,815	2,740	2,737
Tin (Metric Tons)	512	439	504	436	426	419	418	425	401	348	352	347

(*b*) All countries except Belgium and Poland.

The zinc statistics for the last two months of 1931, according to the American Zinc Institute, in tons, follow:

	Nov.	Dec.
Production	20,526	21,965
Production, daily rate	684	709
Shipments	20,327	23,005
Stocks, end of month	130,865	129,825
Retorts operating end of month	21,828	22,275
Retorts, average during month	20,645	20,623

For the entire year 1931, the Institute shows domestic production of 301,073 tons, compared with 504,463 tons of slab produced in 1930. Shipments in 1931 were 314,866 tons; in 1930, 436,275 tons.

Tin Dull

The world's visible supply of tin at the end of December was estimated by the National Metal Exchange at 51,313 long tons, against 50,583 long tons a month previous. The increase in stocks disappointed traders. Tin prices were irregular, tending lower, largely in sympathy with the fluctuations in sterling exchange. Further curtailment in output was put into effect on Jan. 1.

Silver Output Down

World production of silver during 1931 amounted to 196,100,000 oz., according to an estimate by Handy & Harman. This contrasts with an estimated output of 246,800,000 oz. in 1930. The reduced rate of production is expected to have a stabilizing influence on the market. The average price of silver for 1931 was 28.700c. per oz., against 38.154c. in 1930.

Lead and Zinc Concentrates

Galena, basis 80 per cent lead, closed the year with prices more or less nominal at $37.50 to $40 per ton, Joplin, Mo.

Blende, prime, basis 60 per cent zinc, settled at $18 per ton, Joplin, Mo., with the market dull. During 1931 zinc concentrate stocks increased 47,100 tons in the Tri-State district.

Other Metals

Quotations cover wholesale lots, f.o.b. New York, unless otherwise specified. London prices are according to latest mail advices, and at present are largely nominal in view of erratic sterling exchange.

ALUMINUM—Per lb., delivered, Alcoa commercial and mill ingot, 99 and 98 per cent, 23.30c.; Alcoa No. 12 alloy, 22c.; metallurgical ingot, 94 per cent plus, 23.30c.; 98@99 per cent, 22.90c. London, £95, less 2 per cent, per long ton, for 98 per cent ingots and bars.

ANTIMONY—Per lb., duty paid. Chinese, spot, 6.125c.; futures, 5.90@5.95c. The demand has been dull.

BERYLLIUM—See Alloys.

BISMUTH—Per lb., in ton lots, $1. London (dollar basis) 94c.

CADMIUM—Per lb., 55c. London, 2s. 4½d. to 2s. 5½d. Nominal.

CHROMIUM—Per lb., 97@98 per cent grade, 85@90c. per lb. contained chromium, maximum 1 or 2 per cent iron. (Usually sold as ferrochrome.)

COBALT—Per lb.: Metal imported from Belgium, 97@99 per cent, $2.50 less 35 per cent for cash. On yearly requirements, usual rebate of 5 to 10 per cent, as to quantity. London quotes $3.75 per kilo, subject to usual discounts.

INDIUM—Per oz., 98½ per cent, $90@$100 per oz. Nominal.

IRIDIUM—Per oz., $90@$100 for 98@99 per cent sponge and powder. London, £26@£27. Nominal.

LITHIUM—Per lb., 98@99 per cent, 100-lb. lots, $15.

MAGNESIUM—Per lb., ingots (4x16 in.), 99½ per cent, 30c. in carloads; 32c. in 100 lb. lots or more, l.c.l.; ⅜, ½, 1, and 2 lb. sticks, 5c. per lb. over ingot price; ⅛ and ¼ lb. sticks, 8c. per lb. extra.

MANGANESE—Per lb., 95@97 per cent, 42c. (Usually sold as ferromanganese.)

MOLYBDENUM—Per lb., in 10- to 50-lb. lots, C.P. powder, $9; 97 per cent, $4.50. (Usually sold as calcium molybdate or ferromolybdenum, which see.)

NICKEL—Per lb. electrolytic cathodes, 35c.; shot and ingot made from remelted electro, 36c. for single lots of spot metal. London, per long ton, £250 to £255, with prices nominal.

OSMIUM—Per oz., $65@$70. London, £20 10s. to £21.

PALLADIUM—Per oz., $19@$21. London, £5 10s., nominal.

PLATINUM—Per oz. Official price of leading interest, $40. Cash transactions between dealers and refiners several dollars less. London market nominal at £11 to £11 6d. The average price of Colombian crude, basis 85 per cent platinum, for November was $30.25 per oz.

QUICKSILVER — Per 76-lb. flask, $65@$66. Demand continues slow. Mercurio Europeo reduced its price to the basis of $65 f.a.s. European ports.

RADIUM—Per mg. radium content, $50 in lots of 4 grams or more, to $65 for 1 gram; smaller quantities, $70.

RHODIUM—Per oz., $55.

RUTHENIUM—Per oz., $50@$55.

SELENIUM—Per lb., $1.80@$2, depending on quantity, for black, powdered, 99.5 per cent pure.

SILICON—Per lb., minimum 97 per cent Si, maximum 1 per cent Fe, 15@17c.

TANTALUM—Per kilo, $91 for C.P. bar or sheet.

TELLURIUM—Per lb., $2.

THALLIUM—Per lb., $12.50@$15.

TITANIUM—Per lb. in 50- to 100-lb. lots, 75 per cent, $5.

TUNGSTEN—Per lb., contained tungsten, 98 per cent, powdered, $1.45.

Average Metal Prices for December, 1931

COPPER:	
Electrolytic, refinery	6.580
London Standard, Spot	38.273
London Electrolytic, Bid	44.409
LEAD:	
New York	3.792
St. Louis	3.592
London Spot	15.188
London Forward	15.361
SILVER:	
New York	30.120
London	20.023
Sterling Exchange	336.798
ZINC:	
St. Louis	3.149
London Spot	14.361
London Forward	14.818
TIN:	
Straits	21.328
London Standard Spot	138.909
QUICKSILVER	66.115
ANTIMONY	6.231
PLATINUM, Refined	40.000
CADMIUM	55.000
ALUMINUM, 99 Per Cent Plus	23.300

Metallic Ores

Prices in tons of 2,000 lb., or in "units" of 20 lb., unless otherwise stated.

ANTIMONY ORE—No sales reported. London, per long ton unit, 4s. for 60 to 65 per cent sulphide ore.

BERYLLIUM ORE—Per ton, minimum 10 per cent BeO, $20@$30, f.o.b. mines.

CHROME ORE—Per long ton, c.i.f. Atlantic ports, Indian ores, $17.50 for 46@48 per cent Cr_2O_3 ore, and $19.50@$20 for 50@51 per cent ore. London, 80s. for 48 per cent Rhodesian.

COBALT ORE—Per lb. of Co, 12 to 14 per cent grade, 50c., f.o.b. cars, Ontario. Market nominal.

IRON ORE—Per long ton, Lower Lake ports. Lake Superior ores:

Mesabi, non-bessemer, 51½ per cent iron, $4.50. Old Range, non-bessemer, $4.65.

Mesabi, bessemer, 51½ per cent iron, $4.65. Old Range, bessemer, 51½ per cent, $4.80.

Eastern ores, cents per long ton unit, delivered at furnaces: Foundry and basic, 56 to 63 per cent, 10c.

Foreign ores, alongside docks Atlantic ports, cargo lots, cents per long ton unit:

North African and Swedish, low-phosphorus, 8@9c.

Spanish and North African, basic, 56@60 per cent, 7@8½c.

Swedish foundry or basic, 65@68 per cent, 8c.

Newfoundland foundry, 55 per cent iron, 8c. Nominal.

MANGANESE ORE—Per long ton unit of Mn, c.i.f. North Atlantic ports, cargo lots, exclusive of duty: Brazilian, 46@48 per cent Mn, 23c.; Chilean, 47 per cent minimum, 29c.; Indian, 48@50 per cent, 25@26c.; Caucasian, 52@55 per cent, 26c.; South African, 52@54 per cent, 23@24c.; 50 to 52 per cent, 22@24c.; 44 to 46 per cent, 21c.

Per ton in carload lots:

Chemical grades, powdered, coarse

or fine, minimum 80 per cent MnO_2, Brazilian or Cuban, $50 in carloads, to $60 barreled. Javan or Caucasian, 85 per cent minimum, $50@$60. Domestic, 70 to 72 per cent, $43@$50 in carloads, f.o.b. mines.

MOLYBDENUM ORE—Per lb. of contained MoS_2, nominally 45c., delivered Pittsburgh, for 75@85 per cent concentrate. London, per long ton unit, nominal at 36s. to 38s. for 80@85 per cent concentrate.

TANTALUM ORE—Per lb., Ta_2O_5, 70c. for 60 per cent ore. Market nominal.

TIN ORE — No market in United States. London market wholly nominal owing to exchange situation.

TITANIUM ORE — Per gross ton, ilmenite, 45@52 per cent TiO_2, f.o.b. Atlantic seaboard, $10@$12, according to grade and impurities. Low-grade domestic, 32 to 35 per cent, about $7 @$8. Rutile, per lb., guaranteed minimum 94 per cent concentrate, 10c.

TUNGSTEN ORE—Per unit of WO_3, N. Y.: Chinese wolframite, $10.75@ $11, duty paid. Bolivian scheelite, $10.90. Domestic, $10.50@$11.50.

VANADIUM ORE—Per lb. V_2O_5 contained, 27c., f.o.b. shipping point.

ZIRCON ORE—Per ton, 55 per cent ZrO_2, f.o.b. Atlantic seaboard, $40@$45 in 30-ton lots. Crude granular zircon, $70, f.o.b. Suspension Bridge, N. Y.; milled, $90.

Non-Metallic Minerals

Prices received for non-metallic minerals vary widely and depend upon the physical and chemical characteristics of the commodity. Hence the following quotations can serve only as a general guide to the prices obtained by producers and dealers in different parts of the United States for their own product. In the last analysis the value of a particular non-metallic mineral can be ascertained only by direct negotiation between buyer and seller.

Tons of 2,000 lb. unless otherwise noted.

AMBLYGONITE — Per ton, f.o.b. mines, 8@9 per cent Li_2O, $50@$60.

ASBESTOS — Per ton, f.o.b. Quebec mines, tax and bags included: Crude No. 1, $400; crude No. 2, $200; spinning fibers, $100@$125; magnesia and compressed sheet fibers, $100@$125; various grades shingle stock, $50@$65; various grades paper stock, $27.50@$35; cement stock, $15@$20; floats, $10@$15.

Per ton c.i.f. New York: Rhodesian crude No. 1, $300; No. 2, $200. Nominal.

Per ton, c.i.f. New York: Russian crude No. 1, $225; No. 2, $175; No. 3, $125. Nominal.

Per ton, f.o.b. mines, Vermont: Shingle stock, $45; paper stock, $35; cement stock, $20.

BARIUM CARBONATE (Witherite) — Per ton, 90 per cent 300 mesh, $42; 100 mesh, $39.

BARYTES—F.o.b. mines:

California: Crude, $7 per ton.

Georgia: Barytes ore, crude $6@ $6.50 per long ton.

Missouri: Per ton, water ground and floated, bleached, $23, car lots, f.o.b. works. Crude ore, minimum 95 per cent $BaSO_4$, less than 1 per cent iron, $6; 1 per cent iron and 93 per cent $BaSO_4$, $5.50, f.o.b. mines.

BAUXITE—Per long ton: Domestic ore, chemical, crushed and dried, 55 to 58 per cent Al_2O_3, 1.5 to 2.5 per cent Fe_2O_3, $6.50@$8, f.o.b. *Alabama* and *Arkansas* mines. Other grades, 56 to 59 per cent Al_2O_3, 5 to 8 per cent SiO_2, $6@$7.50, f.o.b. *Arkansas* mines. Pulverized and dried, 56 to 59 per cent Al_2O_3, 8 to 12 per cent SiO_2, $10@ $12.25, f.o.b. *Arkansas* mines; abrasive grade, crushed and calcined, 78 to 84 per cent Al_2O_3, $14@$16, f.o.b. *Arkansas* mines.

Per metric ton, foreign, c.i.f. Atlantic ports: Dalmatian, 50 to 55 per cent Al_2O_3, 1 to 3 per cent SiO_2, $4.50@ $6. Istrian, 54 to 57 per cent Al_2O_3, 3 to 5 per cent SiO_2, $5.50@$6.50; French, 56 to 59 per cent Al_2O_3, 2 to 4 per cent SiO_2, $5.50@$7.

BENTONITE—Per ton, carload lots, f.o.b. *Wyoming* mines, dried and crushed, in bulk, $8; in bags, $10, F.o.b. Chicago, selected air-floated, $25.

BORAX — Per ton, carload lots, in bags, crystals, $56; granulated, $50; powdered, $57.50; f.o.b. shipping point.

[1]CELESTITE—Per ton in carload lots, 90 per cent $SrSO_4$, finely powdered, $27.

CHINA CLAY (Kaolin)—Per ton, f.o.b. *South Carolina* mines, crude lump, No. 1, $5@$6; crushed, $7; pulverized, $9@$15.

Florida, washed and crushed, No. 1, $12.50; No. 2, $12.

Delaware, No. 1, $14.50.

Imported English, f.o.b. American ports: lump, $17@$21 in bulk.

DIATOMITE—Per ton, f.o.b. *Nevada,* crude, dried, in bags, $10; minus 40 mesh, $15; 300 mesh, $20; high temperature insulation, $30.

EMERY—Per ton, f.o.b. *New York,* domestic crude ore, first grade, $10. Other American ore, delivered to grinders, per gross ton, $16; Turkish and Naxos ore, $30@$35. F.o.b. *Pennsylvania,* in 350-lb. kegs, Turkish, Khasia, and Naxos grain emery, 6½c. per lb.; American, 3½c.

FELDSPAR — Per ton, f.o.b. *North Carolina,* potash feldspar, 200 mesh, white, $15, in bulk; soda feldspar, $18. F.o.b. *Maine,* potash feldspar, white, 200 mesh, $19, in bulk. Granular glasspar, white, 20 mesh, f.o.b. *North Carolina,* $10.50 in bulk.

Virginia, No. 1, 325 mesh, $17; 200 mesh, $15@$16; 160 mesh, $14@$15; No. 1 glassmakers', $11; enamelers', $12.50.

New Mexico: Crude clean No. 1 potash spar, $4.75; ground, $9.50.

FLUORSPAR — Per ton, f.o.b. *Kentucky* and *Illinois* mines:

Gravel, not less than 85 per cent CaF_2, and not over 5 per cent SiO_2, bulk basis, $13. Foundry lump, 85-5, $15@$17. Ground, 95 to 98 per cent CaF_2, and not over 2½ per cent SiO_2, $30 in bulk; $34 in bags or barrels. *Colorado* mines, 82-5, $10.

Foreign fluorspar, gravel, 85-5, $18.50 per gross ton, duty paid.

FULLER'S EARTH — Per ton, f.o.b. *Colorado,* $18, various finenesses to minus 90 mesh.

F.o.b. *Georgia* or *Florida,* 30 to 60 mesh, $16; 15 to 30, $15; 200 up, $10; 100 up, $7.

GARNET—Per ton, f.o.b. *New Hampshire* mines; concentrate, $40; washed grades, $125.

New York: Adirondack garnet concentrates, $85.

Spanish grades, $60, c.i.f. port of entry.

GILSONITE — Per ton, carload lots, f.o.b. *Colorado.*

Brilliant black, $32.90; seconds (mine run), $25.50.

Selects, $30.50, f.o.b. Utah. Nominal.

GRAPHITE—Per lb., f.o.b. *New York:* Ceylon lump, 6½@7½c.; carbon lump, 3@6c.; chip, 5@6c.; dust, 3@4c.; Madagascar flake, 5@6c.

No. 1 flake, 8@16c.; No. 2, 5½c. upward; fine ground, 55 to 70 per cent carbon, 3c. upward; amorphous, 3c. upward.

Crude amorphous graphite, 15@$35 per ton, according to grade.

GREENSAND — Per ton, f.o.b. cars, *New Jersey:* Screened and bagged, best grade, in carload lots, $20.

GYPSUM—Per ton, f.o.b. mill, crushed rock, *Iowa,* $2.50; *Ohio* and *New York,* $3. Ground, *Ohio,* $4; *Iowa,* $6. Agricultural, $6@$7. Calcined, $6@$9.

IRON OXIDE (See Ocher)—Per lb.: Standard Spanish red, 3@4½c.; domestic earth, 2@4½c.

KAOLIN—See China Clay.

KIESELGUHR—See Diatomite.

LEPIDOLITE—Per ton, $50@$60 for ordinary grades. Nominal.

LIMESTONE—Per ton, f.o.b. shipping points, depending on location, either lump or crushed, 25c.@$1.75.

Agricultural, $1 up to $6 per ton, as to grade and point of shipment.

MAGNESITE—Per ton, f.o.b. *California,* dead-burned, $25. Kiln run, 93 per cent MgO (artificial periclase), $68; 88 per cent MgO, $35. Caustic, 95 per cent MgO, $45; 90 per cent, $40. *Washington:* Dead-burned grain magnesite, $22.

MICA—Per ton, f.o.b. *New Mexico,* scrap, white, $21; off color, $18. Punch, white, for disks, per lb., 10c.; for washers, 8c.

Per ton, f.o.b. *New Hampshire,* roofing mica, $23; snow, $34; 40 mesh white, $40; 60 mesh, $48; 100 mesh, 60; 200 mesh, $75. Clean dry mixed bench and mine scrap, $13.

Per lb., f.o.b. *North Carolina:* Punch, 3@5c.; 1½x2 in., 20@30c.; 2x2, 30@50c.; 2x3, 60@90c.; 3x3, $1 @$1.40; 3x4, $1.40@$1.80; 3x5, $1.50 @$2; 4x6, $2.25@$3; 6x8, $2.50@ $3.75; 8x10, $5@$7.50. These prices apply both to No. 1 and No. 2 mica. Black-stained mica takes a discount of 10 to 25 per cent from this schedule.

White North Carolina mica, 70 mesh, $60 to $80. Biotite, or black mica, $15 a ton, unground.

MINERAL BLACK—(Graphitic shale)—Per ton, 325 mesh, 75 per cent carbon. $22.50, f.o.b. *Virgina.*

MONAZITE—Per ton, minimum 6 per cent ThO_2, $60.

OCHER—Per ton, f.o.b. *Georgia* mines, $17.50 in sacks; $21 in barrels. Buff clay, 98 per cent through 325 mesh, $18.

F.o.b. *Virginia,* dark yellow, 300 mesh, 60 per cent ferric oxide, in jute bags, $19.50.

PHOSPHATE—Per long ton, f.a.s. or f.o.b. mines:

Florida, pebble, f.a.s., for export: 76@77 per cent, $7.15; 75 per cent, $6.40; 74@75 per cent, $6.15; 70 per cent, $4.65; 68 per cent, $4.10.

Florida, pebble, domestic: 76@77 per cent, $6.50; 74@75 per cent, $5.50; 72 per cent, $4.50; 70 per cent, $4.

Tennessee, ground lime phosphate, 80 per cent through 300 mesh, 33 per cent P_2O_5, $12.80 per net ton, bags extra.

PUMICE STONE—Per lb., in barrels, powdered, 2½@4c.; selected lump, 5@7c.

PYRITES—Per long ton unit of sulphur, c.i.f. United States ports, guaranteed 48 per cent sulphur, Spanish, 13c.

POTASH—Per ton, contracts:

	Bags	Bulk
Muriate of potash, 80@85 per cent, basis 80 per cent	$37.15	$35.55
Sulphate of potash, 90@95 per cent, basis 90 per cent	48.25	46.65
Sulphate of potash-magnesia, 48@53 per cent, basis 48 per cent	27.80	26.20
Manure salt, 30 per cent	22.15	19.15
Manure salt, 20 per cent	15.65	12.65
Kainit, 14 per cent	12.70	9.70

Prices are delivered to Atlantic ports. Discounts up to 4 per cent for early shipment. Spot prices, $1.25@$1.50 per ton higher than on contract.

[1]QUARTZ ROCK CRYSTALS—For fusing, all sizes, $250 per ton. Prisms for piezo-electrical and optical use command premium.

SILICA—Per ton, water ground and floated, in bags, f.o.b. *Illinois:* 325 mesh, $16@$40 for 92 to 99½ per cent grades. Dry ground, air-floated, 325 mesh, 92@99½ per cent silica, $14@$30. Glass sand f.o.b. producing plant, $1.25@$5 per ton; molding sand, 50c.@$3.50; blast sand, $1.75@$6. *California:* $5 for quartz and $2.50 for sand.

[1]STRONTIANITE—Per ton, lump in carload lots, minimum 90@92 per cent $SrCO_3$, $100.

SULPHUR—Per long ton for domestic market, $18 f.o.b. Texas mines; $22 for export, Atlantic ports.

TALC—Per ton, carload lots, f.o.b. works, containers included unless otherwise specified:

New Jersey: Soapstone, ground, $10@$12.

New York: Double air floated, short fiber, 200 mesh, $13.75; 325 mesh, $14.75.

Vermont: 99 per cent through 200 mesh, extra white, bulk basis, $8@$8.50; 97 to 98 per cent through 200 mesh, medium white, $7@$7.50. Packing in paper bags, $1 per ton extra.

Virginia: 200 mesh, $4.60@$8.25; 325 mesh, $8.75@$10.50.

TRIPOLI — Per ton, burlap bags, paper liners, minimum carload 30 tons, f.o.b. *Missouri:* Once ground through 40 mesh, rose and cream colored, $15. Double ground through 110 mesh, rose and cream, $17. Air-floated through 200 mesh, rose and cream, $22.50.

Metallic Compounds

ARSENIOUS OXIDE (White arsenic)—Per lb., 4c., delivered, all positions. London market wholly nominal owing to exchange fluctuations.

CALCIUM MOLYBDATE OR MOLYTE—Per lb. of contained Mo, 85c.

COBALT OXIDE—Per lb., black oxide, 70@71 per cent grade, $1.35.

COPPER SULPHATE (Blue Vitriol)—Per lb., in car lots, 3.10c. for either large or small crystals.

SODIUM NITRATE—Per 100 lb., crude natural, in bags ex vessel, Atlantic ports, $1.77.

SODIUM SULPHATE (Salt Cake)—Per ton, bulk, f.o.b. works, $16@$18.

ZINC OXIDE—Per lb. in bags, in car lots: Lead-free, 6½c.; 10, 20 or 35 per cent leaded grades, 6¼c.; French red seal, in bags, 9⅜c.

ZIRCONIUM DIOXIDE—Per lb. in 100- to 500-lb. lots, 87 per cent ZrO_2, commercial, 50c.

Refractories

CHROME BRICK—Per ton, f.o.b. shipping point, $45. Grecian refractory ore, 37@42 per cent, 40c. per unit.

FIRECLAY BRICK—Per M., first quality, $40, *Ohio, Kentucky, Central Pennsylvania;* second quality, $32@$35.

MAGNESITE—Brick, per ton, f.o.b. works, 9-in. straights, $65. Dead-burned grain, $40, f.o.b. Chester, Pa., or Baltimore.

SILICA BRICK—Per M., *Pennsylvania,* $40; *Alabama,* $50@$51; *Illinois* and *Indiana,* $49.

ZIRKITE—Per lb., powdered, 65@70 per cent ZrO_2, 3½c. Brick, straights, 80c.@$1 each.

Rolled Metals

COPPER—Per lb., sheets, hot-rolled, 16⅜c.; wire, f.o.b. mill, 9½c.

LEAD SHEETS—Per lb., full rolled, 7c.; clipped, 7¼c.

MONEL METAL—Per lb., sheets, full finished, 42c.; strip, cold-rolled, 45c.; rods, hot-rolled, 35c.

MUNTZ METAL—Per lb., rods, 12c.; sheets, 15⅜c.

NICKEL—Per lb., sheets, full finished. 52c.; rods, hot-rolled, 45c.

NICKEL SILVER—Per lb., sheets, 10 per cent, 21⅜@21¾c.; 18 per cent, 24⅞@25¼c. Wire and rods, 10 per cent, 24⅞c.; 15%, 28⅜c.; 18 per cent, 32⅜c.

PHOSPHOR BRONZE—Per lb., sheets and rods, 5 per cent tin, 27½c.; wire, 5 per cent, 27⅝c.; 10 per cent, 32⅛c.

ZINC SHEETS—Per lb., 9c., f.o.b. works; ribbon, 8½c.; 7 per cent discount on orders for 18 tons or more.

Alloys

BERYLLIUM-COPPER—Per lb., containing 12.5 per cent of Be, $6.35.

FERROCHROME—Per lb. of contained chromium, 4 to 6 per cent carbon, 65 to 70 per cent chromium, 10c., delivered, on contracts; spot, 10½c. Containing 2 per cent carbon, 67 to 72 per cent chromium, spot, 17c.; maximum 0.10 per cent carbon, 25c., spot; 23½c. contracts.

FERROMANGANESE — Per gross ton furnace: domestic and foreign, 78@82 per cent, $70@$75, contracts, depending on quantity. Spiegeleisen, 19@21 per cent, $26@$27 for spot carloads.

FERROMOLYBDENUM—Per lb. of Mo, f.o.b. shipping point, 50@60 per cent Mo, $1.

FERROPHOSPHORUS—Per gross ton, 18 per cent P, $91; electrolytic, 24 per cent, $122.50, f.o.b. *Alabama* and *Tennessee.*

FERROSILICON — Per gross ton, delivered, 50 per cent, $77.50; 75 per cent, $126; 14@16 per cent, $31, f.o.b. Niagrara Falls, N. Y. Spot shipments $5@$10 per ton higher on 50 and 75 per cent grades.

FERROCARBONTITANIUM — Per ton, $140, f.o.b. producer's plant, carlots.

FERROTUNGSTEN—Per lb. of W contained, 75@80 per cent W, $1@$1.10, f.o.b. works.

FERROVANADIUM—Per lb. of V contained, delivered, $3.05@$3.30.

SILICOMANGANESE — Per long ton, carload contracts, delivered, $105 for maximum 2½ per cent carbon; $115 for maximum 1 per cent carbon.

SILICON ZIRCONIUM—Per lb., 47@52 per cent Si, 35@40 per cent Zr, 16@19c.

ZIRCONIUM FERROSILICON—Per Gross ton, 12@15 per cent Zr, 39@43 per cent Si, $103.50@$108.50.

Iron, Steel, and Coke

IRON—Per gross ton, Valley furnaces: Bessemer, $16@$16.50; basic, $15; and No. 2 foundry, $15.50@$16.

STEEL—Per gross ton, base prices, Pittsburgh, billets and slabs, $28; bars, plates, and structural shapes, per 100 lb., $1.50@$1.60.

COKE—Per ton, Connellsville furnace, $2.25@$2.35. Connellsville foundry, $3.25@$4.25. Birmingham furnace, $3.50@$4.25. Byproduct coke, Ohio and Kentucky (Connellsville basis), $5.50; Buffalo and Detroit, $8@$8.50; Birmingham foundry, $5.

[1]Price furnished by Foote Mineral Company, Philadelphia.

Daily Prices of Metals

For the month ended December 31, 1931

United States Market

Dec.	Electrolytic Copper, Refinery	Straits Tin, New York	Lead, Common New York	Lead, Common St. Louis	Zinc, Prime Western, E. St. Louis
1	6.275	20.875	3.85	3.65	3.125@3.15
2	6.275	21.250	3.85	3.65	3.125
3	6.275	21.000	3.85	3.65	3.125@3.15
4	6.275	20.75@21.00	3.85	3.65	3.15
5	6.275	20.750	3.85	3.65	3.15
7	6.275	20.50@20.625	3.85	3.65	3.15
8	6.275	20.600	3.85	3.65	3.175
9	6.275	20.50@20.70	3.85	3.65	3.15@3.175
10	6.275	20.600	3.85	3.65	3.15
11	6.275	20.650	3.85	3.65	3.15
12	6.025@6.275	21.350	3.85	3.65	3.15
14	6.025@6.275	21.550	3.75	3.55	3.15
15	6.275	21.400	3.75	3.55	3.15
16	6.275	21.600	3.75	3.55	3.15
17	6.575	21.625	3.75	3.55	3.15
18	6.650	21.125	3.75	3.55	3.15
19	7.025	21.625	3.75	3.55	3.15
21	7.025	21.700	3.75	3.55	3.15
22	7.025	21.875	3.75	3.55	3.15@3.20
23	7.025	21.950	3.75	3.55	3.15
24	7.025	22.000	3.75	3.55	3.15
25	Holiday				
26	7.025	22.000	3.75	3.55	3.15
28	7.025	21.875	3.75	3.55	3.15
29	7.025	21.700	3.75	3.55	3.125@3.15
30	7.025	21.700	3.75	3.55	3.125@3.15
31	7.025	21.700	3.75	3.55	3.125@3.15
Av. for Mo.	6.580	21.328	3.792	3.592	3.149
Dec.	Averages for E. & M. J. Week				
2	6.325	21.605	3.850	3.650	3.128
9	6.275	20.731	3.850	3.650	3.154
16	6.233	21.192	3.800	3.600	3.150
23	6.888	21.650	3.750	3.550	3.154
30	7.025	21.855	3.750	3.550	3.145
Dec.	Calendar Week Averages				
5	6.275	21.025	3.850	3.650	3.138
12	6.254	20.727	3.850	3.650	3.156
19	6.492	21.488	3.750	3.550	3.150
26	7.025	21.905	3.750	3.550	3.155

Silver, Gold, and Sterling Exchange New York and London

Dec.	Sterling Exchange "Checks"	Sterling Exchange "90-Day" Demand"	Silver New York	Silver London	Gold, London
1	330.50	329.00	28.500	19.2500	125s ..d
2	340.00	338.00	28.750	19.4375	126s 7d
3	336.75	336.00	28.750	19.1250	122s 3d
4	333.50	332.25	28.750	19.4375	122s ..d
5	329.25	328.00	29.000	19.6250	.....
7	325.50	324.50	30.000	20.5000	126s 2d
8	324.50	324.00	29.625	20.6875	126s 10d
9	327.00	325.50	29.125	19.7500	126s 6d
10	329.75	329.75	29.625	20.1875	125s 3d
11	330.50	330.50	30.125	20.4375	125s ..d
12	339.00	338.25	29.750	20.0625	.....
14	341.50	340.50	30.625	19.8125	118s 9d
15	343.50	343.50	31.250	19.8750	119s 1d
16	344.25	343.75	31.625	20.2500	119s 7d
17	344.50	344.00	31.625	20.4375	119s 6d
18	333.75	333.25	30.750	20.3750	122s 2d
19	339.00	338.50	31.000	20.4375	.....
21	338.50	338.25	30.750	20.4375	122s 1d
22	339.50	339.00	30.625	20.3125	121s 10d
23	341.25	341.25	30.625	19.9375	120s 7d
24	343.25	343.00	30.625	20.0000	120s 2d
25	Holiday				
26	343.00	342.75	30.625	Holiday	
28	341.50	341.00	30.375	20.0000	120s 8d
29	340.50	339.50	30.250	20.0000	120s ..d
30	338.50	338.00	30.125	20.0000	120s 8d
31	338.00	337.00	30.250	20.1875	121s 11d
Av. for Mo.	336.798	.....	30.120	20.023	.....
Dec.	Averages for E. & M. J. Week				
2	342.900	.....	28.925	.....	.....
9	329.417	.....	29.208	.....	.....
16	338.083	.....	30.500	.....	.....
23	339.417	.....	30.896	.....	.....
30	341.350	.....	30.400	.....	.....

Calendar week averages: New York silver: December 5th, 28.688c.; 12th, 29.708c.; 19th, 31.146c.; 26th, 30.650c.

London Market

Dec.	Copper Standard Spot	Copper Standard 3Mo.	Copper Electrolytic	Tin Spot	Tin 3Mo.	Lead Spot	Lead 3Mo.	Zinc Spot	Zinc 3Mo.
1	37.8750	38.5000	43.00	138.375	141.000	15.4375	15.5625	14.5000	15.0000
2	37.3750	38.0000	43.00	137.250	140.000	15.0625	15.2500	14.3750	14.8750
3	36.6250	37.2500	42.00	136.500	139.375	15.0625	15.1875	14.4375	14.8750
4	36.6875	37.3125	42.00	136.375	139.250	15.0625	15.1875	14.3750	14.8125
7	38.0000	38.6250	43.00	138.125	141.000	15.3125	15.4375	14.5000	14.9375
8	38.5000	39.1250	43.00	138.250	141.125	15.5000	15.6250	14.5000	14.9375
9	38.0000	38.6250	43.00	137.250	140.125	15.1250	15.3125	14.1875	14.6250
10	37.5000	38.1250	43.00	136.750	139.750	15.0000	15.2500	14.0625	14.5625
11	37.5000	38.1250	44.00	137.000	140.000	14.9375	15.1875	14.1875	14.6875
14	36.7500	37.3125	43.50	136.875	139.875	14.8125	15.0625	14.0000	14.5000
15	36.6875	37.3125	43.00	136.375	139.375	14.8125	15.0625	14.0000	14.5000
16	37.8750	38.4375	43.00	138.125	141.125	14.8750	15.1250	14.1250	14.6250
17	38.9375	39.5625	43.50	138.500	141.375	15.0000	15.2500	14.3125	14.8125
18	41.1250	41.7500	47.50	138.875	142.000	15.0000	15.2500	14.4375	14.9375
21	40.1250	40.7500	47.00	141.000	144.000	15.3125	15.5625	14.7500	15.1875
22	40.4375	41.1250	47.00	141.875	144.875	15.3750	15.7500	14.6875	15.1250
23	38.5625	39.2500	46.00	141.625	144.750	15.2500	15.5000	14.5625	14.9375
24	39.1250	39.8125	46.00	141.750	144.750	15.2500	15.5000	14.5625	14.9375
25	Holiday								
28	39.1875	39.8750	47.00	141.750	144.875	15.3750	15.4375	14.3125	14.7500
29	38.5000	39.2500	46.00	141.125	144.250	15.6250	15.5625	14.3750	14.8125
30	38.5000	39.1875	46.00	141.125	144.375	15.5625	15.5000	14.3750	14.8125
31	38.1250	38.6875	45.50	141.125	144.250	15.3750	15.3750	14.3125	14.7500
Average for Mo..	38.273	.....	44.409	138.909	.....	15.188	15.361	14.361	14.818

The United States quotations are our appraisal of the major markets for domestic consumption based on sales reported by producers and agencies. They are reduced to the basis of cash, New York or St. Louis, as noted. All prices are in cents per pound.

Copper, lead, and zinc quotations are based on sales for both prompt and future deliveries; tin quotations are for prompt delivery only.

Quotations for zinc are for ordinary Prime Western brands. Zinc in New York is now quoted at 0.35c. per pound above St. Louis, this being the freight differential. Contract prices for High-Grade zinc delivered in the East and Middle West are 1c. above St. Louis prices for Prime Western.

Quotations for lead reflect prices obtained for common lead, and do not include grades on which a premium is asked.

London prices for lead and zinc are the official prices for the morning session of the London Metal Exchange; prices for copper and tin are the official closing buyers' prices. All are in pounds sterling per long ton (2,240 lb.).

New York silver quotations are as reported by Handy & Harman and are in cents per troy ounce of silver, 999 fine. London silver quotations are in pence per troy ounce of bar silver, basis of 925 fine. Sterling quotations represent the demand market in the forenoon. Cables command a premium.

Metal Statistics

Monthly Average Prices of Metals

(See daily price table for basis of quoting)

Silver

	New York		London Spot		Sterling Exchange	
	1930	1931	1930	1931	1930	1931
January	45.000	29.423	20.896	13.810	486.447	485.260
February	43.193	26.773	20.008	12.432	485.784	485.551
March	41.654	29.192	19.298	13.524	485.933	485.596
April	42.428	28.279	19.554	13.120	486.019	485.764
May	40.736	27.650	18.850	12.858	485.724	486.188
June	34.595	27.250	16.049	12.707	485.625	486.291
July	34.346	28.255	15.928	13.197	486.300	485.349
August	35.192	27.524	16.283	12.815	486.839	485.284
September	36.315	28.180	16.738	14.101	485.888	451.245
October	35.846	29.538	16.563	17.153	485.719	388.029
November	35.908	32.223	16.625	19.393	485.454	371.130
December	32.635	30.120	15.201	20.023	485.413	336.798
Year	38.154	28.700	17.666	14.594	485.929	452.707

New York quotations, cents per ounce troy, 999 fine. London, pence per ounce, sterling silver, 925 fine. Sterling exchange in cents.

Copper

	F.O.B. Refinery Electrolytic		London Spot Standard		London Spot Electrolytic	
	1930	1931	1930	1931	1930	1931
January	17.775	9.838	71.469	44.938	83.250	47.524
February	17.775	9.724	71.419	45.372	83.500	47.950
March	17.775	9.854	69.202	44.818	83.405	47.699
April	15.621	9.392	62.075	42.694	74.338	45.375
May	12.756	8.665	53.159	38.897	59.545	42.175
June	12.049	8.025	50.003	35.827	56.750	38.966
July	11.023	7.698	48.277	34.402	52.522	37.293
August	10.693	7.292	47.525	32.572	50.725	35.388
September	10.310	6.988	46.264	31.503	49.500	36.148
October	9.597	6.775	43.030	34.957	45.772	41.000
November	10.113	6.558	46.134	35.854	48.963	41.190
December	10.300	6.580	46.771	38.273	50.065	44.409
Year	12.982	8.116	54.611	38.342	61.528	42.093

New York quotations, cents per pound. London, pounds sterling per long ton

Lead

	New York		St. Louis		London			
	1930	1931	1930	1931	1930 Spot	1930 3 Mos.	1931 Spot	1931 3 Mos.
January	6.250	4.802	6.100	4.604	21.545	21.571	13.872	13.905
February	6.236	4.552	6.086	4.340	21.188	21.097	13.444	13.550
March	5.662	4.527	5.542	4.276	18.807	18.940	13.128	13.355
April	5.537	4.412	5.428	4.164	18.319	18.363	12.375	12.606
May	5.523	3.818	5.408	3.651	17.795	17.861	11.491	11.778
June	5.410	3.917	5.310	3.761	17.941	17.994	11.582	11.952
July	5.250	4.400	5.149	4.225	18.160	18.063	12.731	12.899
August	5.488	4.400	5.340	4.225	18.294	18.178	11.944	11.994
September	5.500	4.400	5.348	4.217	17.909	17.798	11.932	12.026
October	5.151	3.964	4.999	3.773	15.747	15.674	13.227	13.270
November	5.100	3.937	4.950	3.758	15.954	15.931	14.577	14.491
December	5.100	3.792	4.950	3.592	15.283	15.292	15.188	15.361
Year	5.517	4.243	5.384	4.049	18.077	18.064	12.958	13.099

New York and St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Tin and Cadmium

	New York		London		Cadmium
	1930	1931	1930	1931	1931
	Straits		Spot		
January	38.851	26.137	175.466	115.798	67.115
February	38.675	26.315	173.750	117.919	55.000
March	36.798	27.065	164.851	121.852	55.000
April	36.077	25.222	162.638	112.775	55.000
May	32.108	23.221	144.818	104.331	55.000
June	30.336	23.478	136.300	104.966	55.000
July	29.822	24.978	134.511	111.478	55.000
August	30.044	25.738	134.988	114.875	55.000
September	29.647	24.618	132.631	117.813	55.000
October	26.802	22.723	117.451	126.932	55.000
November	25.904	22.779	113.519	132.857	55.000
December	25.262	21.328	111.560	138.909	55.000
Year	31.694	24.467	141.873	118.375	56.010

New York quotations, cents per pound. London, pounds sterling per long ton.

Zinc

	St. Louis		London			
	1930	1931	1930 Spot	1930 3 Mos.	1931 Spot	1931 3 Mos.
January	5.229	4.035	19.634	20.241	12.747	13.113
February	5.180	4.012	19.209	19.778	12.303	12.694
March	4.934	4.002	18.304	18.810	12.190	12.676
April	4.843	3.717	17.819	18.378	11.353	11.838
May	4.641	3.306	16.639	17.330	10.484	10.875
June	4.441	3.416	16.422	17.038	11.270	11.750
July	4.350	3.893	16.171	16.777	12.280	12.802
August	4.360	3.817	15.953	16.469	11.444	12.028
September	4.270	3.744	15.773	16.080	11.571	12.063
October	4.050	3.377	14.446	14.935	12.733	13.216
November	4.266	3.209	14.706	15.238	13.845	14.247
December	4.099	3.149	13.762	14.214	14.361	14.818
Year	4.556	3.640	16.570	17.107	12.215	12.677

St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Antimony, Quicksilver, and Platinum

	Antimony (a) New York		Quicksilver (b) New York		Platinum (c) New York	
	1930	1931	1930	1931	1930	1931
January	8.606	7.317	121.192	103.000	61.923	36.000
February	8.830	7.069	120.500	100.205	59.909	34.000
March	8.236	7.127	118.808	100.423	54.769	29.538
April	7.740	6.888	114.000	102.077	46.462	26.346
May	7.454	6.524	117.269	101.140	46.000	24.980
June	7.057	6.342	118.600	92.058	46.000	37.115
July	6.976	6.802	117.462	85.808	43.692	40.000
August	7.839	6.596	116.000	80.115	41.462	40.000
September	7.773	6.542	113.620	76.300	36.080	40.000
October	7.288	6.517	110.596	72.385	36.000	40.000
November	7.105	6.679	106.717	68.587	36.000	40.000
December	7.099	6.231	105.346	66.115	36.000	40.000
Year	7.667	6.720	115.009	87.351	45.358	35.665

(a) Antimony quotations in cents per pound, for ordinary brands. (b) Quicksilver in dollars per flask of 76 lb. (c) Platinum in dollars per ounce.

Pig Iron[1] and Aluminum

	Bessemer		Basic		No. 2 Foundry		Aluminum	
	1930	1931	1930	1931	1930	1931	1930	1931
January	19.00	17.50	18.50	17.00	18.50	17.00	24.300	23.300
February	19.00	17.29	18.50	16.79	18.50	16.79	24.300	23.300
March	19.00	17.00	18.50	16.50	18.50	16.50	24.300	23.300
April	19.00	17.00	18.50	16.50	18.50	16.50	24.300	23.300
May	19.00	17.00	18.50	16.38	18.50	16.44	24.300	23.300
June	19.00	17.00	18.50	15.50	18.50	16.00	24.140	23.300
July	18.62	17.00	18.12	15.50	18.12	16.00	23.300	23.300
August	18.50	17.00	18.00	15.50	18.00	16.00	23.300	23.300
September	18.38	17.00	17.76	15.50	17.88	16.00	23.300	23.300
October	17.85	16.92	17.00	15.42	17.22	16.00	23.300	23.300
November	17.50	16.50	17.00	15.00	17.00	16.00	23.300	23.300
December	17.50	16.25	17.00	15.00	17.00	15.75	23.300	23.300
Year	18.53	16.96	17.99	15.88	18.02	16.25	23.787	23.300

Iron in dollars per long ton. Aluminum in cents per pound, 99 per cent grade
[1]F.o.b. Mahoning and Shenango Valley furnaces; freight to Pittsburgh, $1.76

Monthly Crude Copper Output in Short Tons

	1930	1931 July	1931 Aug.	1931 Sept.	1931 Oct.	1931 Nov.
Domestic						
Alaska shipments	18,953	1,048	748	2,090	338	214
Magma	15,940	1,229	.....	.....	.....	.....
Miami	34,568	2,032	.....	.....	.....	.....
Nevada Consolidated	70,990	.....	.....	(d) 16,194	.....	.....
Old Dominion (a)	10,428	575	539	.....	.....	.....
Phelps Dodge (b)	72,308	5,179	5,166	.....	.....	.....
United Verde Extension	21,908	.....	.....	.....	1,642	1,392
Tennessee Corporation	7,772	741	.....	.....	.....	.....
Foreign						
Andes Copper	47,428	3,476	3,498	.....	.....	.....
Braden	79,923	8,595	8,593	.....	.....	.....
Boleo, Mexico	13,940	.....	.....	(d) 3,283	.....	.....
Chile	89,100	7,121	7,115	.....	.....	.....
Furukawa, Japan	18,536	1,431	1,427	1,561	1,683	.....
Howe Sound	22,633	.....	.....	.....	.....	.....
Messina	(c) 7,600	832	897	860	873	815
Mt. Lyell	10,878	.....	.....	.....	766	692
Sumitomo, Japan	15,429	1,255	1,260	955	.....	.....

(a) Includes Arizona Commercial. (b) Moctezuma is included. (c) Fiscal year. (d) Three months.

Monthly Production of Primary Copper from U. S. Mines and Daily Rate (Short Tons)

	1929 Monthly Production	1929 Daily Rate	1930 Monthly Production	1930 Daily Rate	1931 Monthly Production	1931 Daily Rate
January	86,324	2,785	67,838	2,188	48,059	1,550
February	84,735	3,026	59,196	2,144	47,504	1,697
March	93,698	3,023	61,216	1,975	48,702	1,571
April	94,902	3,163	60,450	2,015	46,452	1,548
May	93,392	3,013	60,238	1,943	45,580	1,470
June	82,354	2,745	56,743	1,891	44,473	1,482
July	79,229	2,556	54,249	1,750	38,228	1,233
August	78,885	2,545	56,779	1,832	38,925	1,256
September	79,402	2,647	56,584	1,886	38,088	1,270
October	82,575	2,664	55,954	1,805	.....	.....
November	75,934	2,531	53,141	1,771	.....	.....
December	74,772	2,412	48,518	1,565	.....	.....
Total	1,006,203	....	690,263	....	396,011	....
Monthly average	83,850	....	57,522	....	44,001	....
Av. of daily rate	.....	2,757	.....	1,891	.....	1,451

Mining Share Prices—Month of December, 1931

Company Name	High	Low	Last	§Most Recent Dividend
NEW YORK STOCK EXCHANGE				
Alaska Juneau	14¼	10⅛	13¼	Ja. 9, Fe. 9, 1932 Q 0.12½
American Metal	7.00	4.87½	5.50	December, 1930 Q 0.25
Am. Metal, pfd.	26	14	16	Au. 22, Se. 1, 1931 Q 1.50
Am. Sm. & Ref.	24½	17½	17¾	Ja. 15, Fe. 1, 1932 Q 0.12½
Am. Sm. & Ref. 2d pfd.	59⅞	45	47	Fe. 5, Mh. 1, 1932 Q 1.50
Am. Sm. & Ref. 1st pfd.	99½	75	78¼	Fe. 5, Mh. 1, 1932 Q 1.75
Am. Zinc, L. & S.	3.75	2.75	2.75	May, 1917 1.00
Am. Zinc, L. & S., pfd.	26	19½	21	October, 1930 Q 1.50½
Anaconda	14½	9.25	9.50	Jy. 11, Au. 17, 1931 Q 0.37½
Andes Copper	5.50	4.00	4.12½	Ja. 10, Fe. 9, 1931 Q 0.25
Butte C. & Z.	1.62½	1.00	1.00	December, 1929 0.50
Butte & Superior	0.87½	0.75	0.75	December, 1929 Q 0.50
Callahan Zinc-Lead	0.37½	0.25	0.25	December, 1920 0.50
Cal. & Hecla	4.00	3.12½	3.25	June, 1930 0.50
Cerro de Pasco Copper	15½	11¼	11½	Oc. 16, No. 1, 1931 Q 0.25
Dome Mines	8.62½	7.12½	7.75	De. 31, Ja. 20, 1932 Q 0.25
Federal M. & S.	36½	22	26	June, 1927 10.00
Federal M. & S., pfd.	40	36	36	My. 25, Je. 5, 1931 Q 1.75
Freeport Texas	17⅛	14⅛	16	No. 15, De. 1, 1931 Q 0.75
Granby Con	7.00	5.25	5.50	Ja. 15, Fe. 1, 1932 Q 0.12½
Greene Cananea	17	13½	17	July, 1930 0.75
Homestake	130½	120	129	De. 19, De. 26, 1931 Q 0.65
Howe Sound	16	11½	13	De. 31, Ja. 15, 1932 Q 0.50
Inspiration	4.00	3.00	3.25	July, 1930 0.50
International Nickel	9.25	7.00	7.50	De. 1, De. 31, 1931 Q 0.05
Int. Nickel, pfd.	88	80	82	Ja. 2, Fe. 1, 1932 Q 1.75
Kennecott Copper	13½	9.62½	10½	De. 17, Ja. 2, 1932 Q 0.12½
McIntyre Porcupine	16	14	13⅝	No. 2, De. 1, 1931 Q 0.25
Magma Copper	8.50	7.37½	8.00	De•31, Ja. 15, 1932 Q 0.12½
Miami Copper	3.87½	2.75	2.75	August, 1930 0.375
Mother Lode	0.37½	0.25	0.25	June, 1930 0.10
National Lead	91½	78½	85	De. 11, De. 31, 1931 QX 1.50
National Lead, pfd. A	130	112	115	No. 27, De. 15, 1931 Q 1.75
National Lead, pfd. B	103	100	100	Ja. 15, Fe. 1, 1932 Q 1.50
Nevada Con.	6.12½	4.75	5.37½	De. 17, De. 31, 1931 Q 0.10
Noranda Mines	15½	11	12¼	De. 15, De. 29, 1931 0.50
Park Utah Con.	1.50	1.25	1.25	July, 1929 Q 0.20
Patiño M. & E.	7.50	5.75	6.12½	December, 1929 4s.
Phelps Dodge	8.00	5.87½	6.75	Je. 18, Jy. 1, 1931 Q 0.25
Rand Mines	22	20	20	August, 1930 1.52
St. Joseph Lead	12½	7.50	7.75	Mh. 10, Mh. 21, 1932 Q 0.15
Seneca Copper	0.50	0.25	0.25	
Tennessee Corp.	3.12½	2.00	2.25	Fe 28, Mh. 16, 1931 Q 0.125
Texas Gulf Sulphur	27⅞	19½	22½	De. 1, De. 15, 1931 Q 0.75
U. S. S. R. & M.	18½	13½	15½	De. 31, Ja. 15, 1932 Q 0.25
U. S. S. R. & M., pfd.	40	37	37½	De. 31, Ja. 15, 1932 Q 0.87½
Utah Copper	55	54	54	De. 17, De. 31, 1931 Q 1.00
Vanadium Corp.	17	11	12½	Fe. 2, Fe. 16, 1931 Q 0.75
NEW YORK CURB EXCHANGE				
Aluminum Co. of Am.	71½	48	50½	
Alum. Co. of Am., pfd.	75½	56½	63	Mr. 14, Ap. 1, 1931 Q 1.50
Anglo-Chilean Con.	0.75	0.25	0.25	4 shares Cosach B. June, 1931
Bunker H. & S.	27½	20½	24	My. 29, Je. 5, 1931 0.25
Carnegie Metals	1.62½	1.00	1.62½	
Comstock Tunnel	0.43¾	0.12½	0.37½	
Con. Coppermines	1.25	0.75	0.87½	
Copper Range	2.50	2.50	2.50	July, 1930 0.25
Cresson Consolidated	0.31½	0.12½	0.12½	Je. 30, Jy. 10, 1931 Q 0.01
Cusi Mexicana	0.56½	0.37½	0.43½	
Evans-Wallower	0.56½	0.12½	0.62½	
Federated Metals	7.50	5.87½	6.00	Ja. 5, Ja. 15, 1931 Q 0.25
Hecla Mining	5.50	4.25	4.25	No. 15, De. 15, 1931 Q 0.10
Hollinger Consolidated	5.00	4.00	4.75	De. 16, De. 31, 1931 FW 0.05
Hudson Bay M. & S.	2.62½	1.75	2.00	
Iron Cap Copper	0.62½	0.50	0.62½	
Mayflower Associates	30	25½	28	De. 1, De. 15, 1931 Q 0.50
Mining Corporation	1.12½	1.00	1.00	December, 1929 0.12½
New Jersey Zinc	29	20½	22	Ja. 20, Fe. 10, 1932 Q 0.50
Newmont Mining	16	9.12½	10	Mh. 31, Ap. 15, 1931 Q 1.00
N. Y. & Honduras Ros.	13¾	13	13½	De. 10, De. 26, 1931 X 0.50
Nipissing	1.12½	0.87½	0.93¾	Mh. 31, Ap. 20, 1931 Q 0.075
Ohio Copper	0.18¾	0.06½	0.12½	September, 1926 0.03
Premier Gold	0.62½	0.43¾	0.43¾	De. 14, Ja. 4, 1932 Q 0.03
Quincy	2.25	2.00	2.00	March, 1920 1.00
Shattuck Denn	2.25	1.50	1.62½	
So. Am. Gold & Plat.	0.87½	0.50	0.75	
Teck-Hughes	4.62½	3.50	4.00	Ja. 16, Fe. 1, 1932 Q 0.15
Tonopah Mining	0.37½	0.37½	0.37½	Octobe 1929 0.07½
United Eastern			0.06¼*	
United Verde Ex.	4.37½	3.12½	4.25	Ja. 2, Fe. 1, 1932 Q 0.25
Utah-Apex	0.43¾	0.43¾	0.43¾	Decem er, 1929 0.02
Yukon Gold	0.18¾	0.12½	0.18½	June, 1918 0.02
BOSTON STOCK EXCHANGE				
Isle Royale	2.00	1.25	1.50	March, 1930 0.50
Mohawk Mining	17	11¾	11¾	Oc. 31, No. 28, 1931 Q 0.25
Old Dominion	1.25	0.90	0.90	December, 1918 1.00
Utah Met. & Tunnel	0.39	0.33	0.35	December, 1927 0.30
¶MONTREAL				
Asbestos Corp.	0.30	0.25	0.25	January, 1926 1.50
Asbestos Corp., pfd.	2.00	2.00	2.00	January, 1930 1.75
Consolidated M. & S.	75	66	66	De. 23, Ja. 15, 1932. SAX 1.25

Company Name	High	Low	Last	§Most Recent Dividend
SPOKANE				
Constitution	....	....	0.06¼*	
Dayrock	....	....	0.95*	March, 1930 0.03
Golconda	0.24	0.19½	0.19½	July, 1929 0.02
Jack Waite Con.	0.27	0.21½	0.22½	
Los Lugos	....	....	0.01½*	
Mexican Premier	0.08¼	0.06½	0.08	January, 1930 0.01
Montana Mines	....	....	0.09	
Noble Five	....	....	0.03*	January, 1930 0.005
Pend Oreille	0.95	0.85	0.85	
Sherman Lead	0.27½	0.27½	0.27½	March, 1930 0.01
Sidney	....	....	0.05	October, 1929 0.01
Sunshine Mining	1.35	0.98	0.98	De. 10, De. 23, 1931 X 0.02
Tamarack & Custer	0.17	0.16	0.16	September, 1924 0.25
SALT LAKE CITY				
Chief Con.	42	38½	41	February, 1928 0.10
Combined Metals	7½	6½	7½	
Eureka Bullion	20	15½	18½	
Eureka Lilly	13	11	11	
Moscow Silver	....	....	0.40	
New Quincy	5½	4½	4½	September, 1929 Q 0.10
North Lily	81	71	71	January, 1930 0.15
Park Bingham	2½	1½	1½	
Park City Con.	14½	10½	12	
Rico Argentine	4	3½	3½	December, 1929 0.03
Silver King Coalition	4.05	3.27½	3.60	De. 20, Ja. 2, 1931 Q 0.15
Tintic Lead	98	85	95	
Tintic Standard	3.75	2.85	3.00	De. 10, De. 24, 1931 Q 0.10
Walker Mining	91	66	85	April, 1930 0.075
¶TORONTO				
Abana Mines	0.05	0.03	0.04	
Amulet Mines	0.22	0.16½	0.22	
Barry-Hollinger	1.00	0.85	0.95	
Base Metals	0.09¾	0.08	0.09	
Canam Metals	....	....	0.02*	
Castle-Trethewey	0.19	0.14	0.16	
Central Manitoba	0.08	0.06	0.07	
Falconbridge	1.50	1.00	1.00	
Granada Gold	1.50	1.09	1.09	
Howey Mines	0.33½	0.30	0.32	
Keeley	0.30	0.21	0.28	March, 1928 0.04
Kirkland Lake	0.62	0.38	0.51	
Lake Shore	28.10	26.00	27.50	De. 1, De. 15, 1931 QX 1.00
Sherritt Gordon	0.68	0.58	0.65	
Siscoe Gold	0.60	0.45	0.52	
Sylvanite	0.65	0.55	0.58	De. 5, De. 31, 1931 SA 0.02
Treadwell Yukon	1.25	0.90	0.90	
Ventures	0.51	0.41	0.48	
Vipond Con.	0.54	0.37	0.42	April, 1927 0.03
Waite-A.-M.	1.00	0.75	0.80	
Wright-Hargreaves	3.10	2.62	2.72	De. 14, Ja. 2, 1932 Q 0.05
ST. LOUIS, CINCINNATI, AND DENVER				
Consolidated L. & Z.	1.50	1.50	1.50	December, 1929 0.125
Eagle-Picher	5.25	4.87½	5.00	April, 1930 0.20
Eagle-Picher, pfd.	....	....	89*	De. 31, Ja. 15, 1931 Q 1.50
Empire Chief	0.01	0.00¾	0.00¾	
Star of Utah	0.18	0.15	0.15	
Tungsten Prod.	0.10	0.07	0.09½	
Ute Gold	0.25	0.15	0.15½	
SAN FRANCISCO, LOS ANGELES, AND COLORADO SPRINGS				
California Copper	0.25	0.12½	0.25	
(a) Golden Cycle	12	10	11	No. 30, De. 10, 1931 Q 0.40
(a) Granite Gold	0.20	0.15	0.19	De. 23, Ja. 2, 1932 Q 0.01
Idaho-Maryland	1.64½	1.28	1.46	
Tom Reed	0.54	0.47	0.51	De. 15, Ja. 1, 1932 Q 0.03
(a) United Gold	0.05½	0.04	0.05	Oct. 15, No. 1, 1931 0.01

‡LONDON

Company Name	High	Low	Last	Most Recent Dividend (Per Cent, Unless Noted)
Anglo American	9/9	7/6	7/9	22½, Feb. 19. 1930
Aramayo Mines	16/10½	15/–	15/–	5, Oct. 23, 1929
Burma Corporation	9/4½	7/10½	8/7½	1¼, Dec. 10. 1930(b)
Camp Bird	–/5	–/3	–/4½	16¼, April 25, 1928
Fresnillo	9/–	9/–	9/–	25 cents, Feb. 12, 1930
Frontino & Bolivia	17/6	14/–	15/–	2½, July 1, 1931
Lake View & Star	9/–	7/6	7/9	11¼, 1921
Mexican Corporation	4/–	3/–	3/6	10, Oct. 24, 1929
Oroville Dredging	1/6	1/6	1/6	37½, May 16, 1930
Rhokana Corporation	85/–	55/7½	65/–	
Rhodes.-Select. Trust	7/6	6/–	6/6	
Rio Tinto	17½	12	12½	10s, May 1, 1931
St. John del Rey	19/6	17/9	18/6	9d, No. 20, 1931
San Francisco Mines	10/1½	7/9	8/6	10, Jan. 7, 1931
Santa Gertrudis	8/5½	6/3	7/–	9d., July 9, 1931
Selukwe	2/–	1/10	2/–	6¼, April 2, 1917
So. American Copper	–/10½	–/5	–/6	
Tanganyika	18/1½	14/3	16/10½	2½, Aug. 2, 1930
Wiluna Gold	9/9	8/–	8/–	
Union Minière	3,250	3,000	3,085	240 francs, July, 1931

Salt Lake Stock Exchange courtesy J. A. Hogle & Company. ¶These figures are quoted in Canadian dollars. *Not traded in. †Belgian Francs. ‡The above London quotations are for the month ended Dec. 24. Q, quarterly; I, initial; M, monthly; X, extra; SA, semi-annual; L, liquidating B, plus bonus. §Total dividends paid during the month will be found tabulated in the Market and Financial News Section. Dates here given are those of closing of books and payment of dividend. (a) Bid prices. (b) 1¼% (2 annas) dividend, and bonus of ¼% (1 anna). (c) Closing bid prices. F.W., four weeks.